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CANADA
DEPARTMENT OF MINES
HON. CHARLES STEWART, Minister CHARLES CAMSELL, Deputy Minister

MINES BRANCH
JOHN McLEISH, Director

SUMMARY REPORT

ON

MINES BRANCH INVESTIGATIONS

DURING THE CALENDAR YEAR
ENDING DECEMBER 31,

1922

OTTAWA
F. A. ACLAND
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
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SUMMARY REPORT OF MINES BRANCH INVESTIGATIONS, 1922

MINERAL RESOURCES DIVISION

I

MINERAL PIGMENTS

Howells Fréchette

In continuation of the investigation on mineral pigments in eastern Canada a number of deposits were visited and examined in Ontario, Quebec and New Brunswick. In all cases these deposits had been recently brought to the attention of the Department through samples submitted for examination.

Ontario

Northern Ontario.—On the east bank of the Abitibi river, on lot 27, concession I, Kennedy township, about 8 miles east of Cochrane, springs issuing from the margin of a sand flat deposit iron ochre over an area of about six acres. Over the greater part the deposit is too thin to be of any commercial interest, but at a few points it reaches a depth of two feet or more. Carbonate of lime has been deposited along with the iron oxide in certain sections of the area, though much of the deposit is sufficiently pure for the production of calcined red oxide for paint manufacture. In view of the shallowness of the deposition, and the admixture of carbonate of lime in places, the deposit cannot be regarded as a potential producer of iron ochre for paint manufacture, as the tonnage of good material is insufficient to warrant the installation of a plant to prepare it for the market. A limited amount of crude oxide could be produced for use in purification of gas should market conditions be favourable.

Probably the most important aspect in which this and other smaller deposits just outside the town of Cochrane should be viewed is as an indication of favourable conditions having existed for the laying down of such deposits. Search along the lower margins of sand areas, especially those which have been subject to the leaching action of swamp water, may disclose larger deposits at present hidden by moss or other vegetable growth.

Central Ontario.—A small group of deposits of iron ochre were examined on lot 29, concession XV, Monmouth township, Haliburton county, about two miles west of Wilberforce and half a mile north of the railway.

These deposits, unlike those near Cochrane, evidently owe their origin to iron leached from rock in place, probably from pyrite occurring in quartzite or amphibolite, for the sand and gravel in the immediate neighbourhood was insufficient to account for the amount of iron ochre deposited.

The deposits lie in a small ravine and are in the form of mounds, built up around springs, which flatten out into thin layers merging into one another. The largest mound covers about 1,000 square feet and rises to a height of at least six feet above the general level of the ravine. Owing to the presence of layers of hard bog ore the bottom of the deposit could not be reached with the sampling drill, and the exact depth determined. Deposition at this point has ceased, the spring having found a new outlet about fifty feet to the west. Here a mound is in course of formation and has reached a thickness of four feet. There are other lesser mounds. In all about one acre is covered with the ochre, though so thinly in most places that it could not profitably be dug. While a considerable tonnage of iron ochre, suitable for calcining for the production of red oxide, could be obtained from this group of deposits, it is insufficient to warrant the erection of a plant. The material is suitable for gas purification and it might be possible to market it profitably for that purpose.

Quebec

Drummond County.—At one time quite a flourishing industry in digging bog iron ore was carried on in this county, but with the disappearance of the small blast furnace no further attention was paid to the deposits. In some cases, associated with the hard bog ore there was a considerable proportion of the earthy iron oxide or iron ochre. No attempt was made to save this for the production of paint except merely for local use. One of the deposits, on lot 11, range II, Simpson township, was examined and found to consist largely of a good grade of iron ochre suitable for the production of calcined red oxide, but the deposit is shallow and irregular as to depth and very limited in area. The district, however, offers a reasonable amount of promise for systematic prospecting, as indications of ochre are seen for many miles both on the north and south sides of the St. Francis river.

St. Rémi d'Amherst.—A few hundred feet to the southeast of the present workings of the Canadian China Clay Co., Ltd., on lot 6, range VI south, Amherst township, Labelle county, there is a showing of yellow ochre. The ochre in this case is of the clay-base type, that is, the material is a clay stained yellow by iron oxide and not simply an earthy hydrated oxide of iron precipitated from a water solution by oxidation such as iron ochre. It corresponds to the French yellow ochre of commerce and produces a paint which compares favourably as to colour and tinting power with that pigment. At the point where this is exposed, along the edge of a railway cutting, there is an overburden of gravel three to four feet thick, which would have to be trenched through in order to determine the tonnage available. The attention of the management of the company was called to the value of the material as a pigment, and to the fact that it was the first example of which we have record of material of this grade in eastern Canada. Later samples of similar material were received from New Brunswick, and the deposits from which they were obtained were examined.

New Brunswick

A number of samples of rather highly coloured clays were submitted by interested parties in St. John, to be examined as possible raw material for the manufacture of paints. They consisted of red, buff and yellow clays, which, when ground in oil, produced paints of poor quality as to opacity, intensity of colour and brilliance, except in the case of the yellow clay, which may be classed as a fairly high grade yellow ochre. The other colours were unsuitable as pigments, except possibly as cheap fillers for linoleum manufacture.

Opportunity offered to visit the deposits from which the above mentioned samples were obtained and an examination was made of them in October.

Northumberland County.—About six miles southeast of Howard station, to the west of Cain river, between Otter brook and the mouth of Sabbies river, there are several exposures of coloured clay. In the bed of a small creek in the southern part of the area indicated above, an unctuous blue-grey clay outcrops from under about five feet of gravel. Within this bluish clay are streaks and small pockets of bright yellow ochre of good grade, but as the quantity apparently is very small, and also due to its mixture with the other clay, it is of no commercial value.

To the northward there are a number of clay areas many acres in extent. The general colour of this clay is buff, though the upper half foot is frequently of a stronger yellow colour. None of it is suitable for paint manufacture.

Kings County.—Clays resulting from the weathering of Carboniferous shales are exposed along the sides and floor of a small valley on the road from St. John to Smithtown, near Damascus. The prevailing colour of the clay is red, but it contains streaks and small patches of greyish white silty clay and yellow clay.

The red clay is similar to much occurring throughout New Brunswick and Nova Scotia. It cannot be regarded as valuable for pigment purposes as it lacks brilliance of colour and opacity when ground in oil. It might be used as a filler in the manufacture of linoleum. The white silty material is worthless. The yellow clay would make good paint similar to French yellow ochre, but it occurs in too small quantity to be of any value, and the indications do not point to the likelihood of more extensive deposits.

II

ALKALI DEPOSITS, WESTERN CANADA

L. H. Cole

INTRODUCTORY

The examination of the alkali lakes of western Canada, begun during the field season of 1921, was continued during the summer of 1922. The work was hindered to some extent during the entire season by the unusual amount of rainfall which occurred in the localities in which the lakes are situated. None of the deposits examined were entirely dry at any time during the work, and most of the drilling was carried on with from six inches to one foot of brine on top of the crystal bed.

The methods employed in the examination were the same as those of the previous season, a description of which is given in the Summary Report of the Mines Branch for 1921. The work was commenced at Vincent lake, 12 miles northwest of Tompkins, Sask., and the season was spent in the area lying to the west of Swift Current, Sask., between the main line of the Canadian Pacific railway and the Empress branch of the same line.

SODIUM SULPHATE DEPOSITS

Three large deposits were drilled and sampled and preliminary work was done on a dozen other deposits.

The deposits examined were very similar to those previously studied. The thickness of the crystal bed varied greatly in the several deposits, and also from place to place in the same lake. One drill hole cut through 30 feet of solid crystals before encountering the underlying mud. This is the greatest depth of crystals so far proven by the investigation.

The drilling done to date on six deposits has proven the presence of 17,000,000 tons of crystals, 90 per cent of which can be considered hydrous sodium sulphate.

Description of Deposits

The following brief description of the deposits examined during the season is only preliminary and will be supplemented in the final report.

Vincent Lake, Sask.

This lake, situated in townships 14 and 15, range 22, west of the 3rd meridian, occupies an undrained basin surrounded by morainic hills which rise in places some 50 feet above the normal level of the lake. There is a crystal bed of varying thickness, 30 feet of crystals being encountered in one of the drill holes. The crystal bed covers an area of approximately 325 acres with an average depth of 10 feet.

The following analyses* are of material from this deposit:—

	Hole No. 7	Hole No. 11	Hole No. 17
Insoluble	6·00%	6·00%	11·00%
Iron oxide and alumina	0·70	0·60	1·23
Calcium carbonate	....	....	2·76
Calcium sulphate	1·94	1·57	....
Magnesium carbonate	....	....	1·55
Magnesium sulphate	2·92	2·29	....
Sodium chloride	0·74	0·77	0·82
Sodium sulphate	86·51	87·71	82·08

*Analyses made on dried material.

This deposit is approximately 11 miles northwest of Tompkins, Sask., on the Canadian Pacific railway.

Chain Lake, Sask.

This is a long narrow lake situated in sections 10, 15, 16 and 21, tp. 18, range 20, west of the 3rd meridian. The surrounding country is comparatively flat-lying and the depression is correspondingly shallow. There is a crystal bed with an average depth of 2·25 feet.

The crystal bed covers approximately 70 acres.

An average analysis of the dried material from this lake is as follows:—

	Per cent
Insoluble	7·75
Iron oxide and alumina	2·00
Calcium carbonate	2·05
Calcium sulphate	3·00
Magnesium sulphate	4·48
Sodium sulphate	79·50

The deposit lies approximately 16 miles to the southwest of Cabri, Sask., a station of the Swift Current-Empress branch of the Canadian Pacific railway.

The water in another lake lying one-half mile to the south is strongly alkaline but no crystal bed was encountered in it.

Snakehole Lake, Sask.

This lake is situated on sections 11, 12, and 13, tp. 18, range 19, west of the 3rd meridian, and is 7 miles south of Cabri station on the Canadian Pacific railway. There is a large crystal bed in the bottom of this lake covering approximately 460 acres. A depth of 20 feet of crystals was encountered in one hole. Numerous springs feed into this lake. At the north end of the depression the banks rise almost vertically in places to a height of 100 feet above the lake level, the slopes of the surrounding hills being more gentle towards the southern end of the deposit. Scattered through the crystal bed are numerous mud holes varying from a few feet to many feet in diameter.

An average analysis of the dried material from this deposit is as follows:—

	Per cent
Insoluble	4·00
Iron oxide and alumina	0·55
Calcium sulphate	2·06
Magnesium sulphate	5·24
Sodium chloride	0·75
Sodium sulphate	87·44

Wiseton, Sask.

There are two lakes 21 miles south of Wiseton, Sask., situated in section 32, tp. 23, range 12, west of the 3rd meridian. These lakes cover approximately 270 acres but there was no crystal bed visible.

An average sample of the brine from these two lakes gave the following results in parts per million:—

Insoluble	20
Calcium sulphate	20
Magnesium sulphate	200,000
Sodium chloride	9,500
Sodium sulphate	80,000

Operating Deposits

Three deposits are being developed at the present time and plans are under way for the opening up of several others.

Salts and Chemicals, Ltd.

At Dana, Sask., a half-million dollar plant is nearing completion, to extract salt cake and other chemical products from the brines of Muskiki lake, formerly Houghton lake, 23 miles west of Humboldt, Sask. The

company operating this deposit is The Salts and Chemicals, Ltd., controlled by Canadian and United States interests. The refining plant of this company, which was formerly at Kitchener, Ont., is now being removed to the lake. When this plant is in operation it is estimated that the output will be 30,000 tons of salt cake per year, besides Epsom and other salts recovered as by-products. A small village has been established at the lake with housing accommodation for 60 men. When the plant is working at fuller capacity 100 men are to be employed.

Bishopric and Lent Co.

At Frederick lake, five miles southwest of Dunkirk, Sask., and 38 miles from Moose Jaw, Sask., another salt cake recovery plant has been erected by The Bishopric and Lent Co., with head offices at Cincinnati, Ohio. This plant is practically completed, and as soon as it proves commercially successful, further units are to be added to increase the output. At the present time 20 men are employed at this plant.

Soda Deposits, Ltd.

At a lake five miles north of Fusilier, Sask., an experimental plant has been operated by a company called Soda Deposits, Ltd., with head office at Calgary, Alta. This deposit, although somewhat smaller than some of the other deposits, contains sodium sulphate in a very pure state, and several shipments have already been made. This company hopes in the near future to be in a position to put its product regularly on the market.

MAGNESIUM SULPHATE AND SODIUM CARBONATE DEPOSITS

In British Columbia, several deposits of magnesium sulphate were visited, as well as a number of occurrences of sodium carbonate.

Spotted Lake, B.C.

This lake lies approximately one-half mile north of the southern entrance to Richter's pass. The nearest town to the deposit is Oroville, Washington, U.S.A., which is distant about 6 miles. It is 40 miles by road south of Penticton, B.C. The deposit occurs in a flat depression known as Spotted lake, which is a partially dried up lake containing alternate circles of saturated brine and dried silt.

A small shipment was made from this lake in 1915, and it has been worked intermittently since that time. Approximately 1,600 tons of crude magnesium sulphate were removed from this deposit up to 1920, when operations apparently were abandoned. This material was hauled by auto truck to the Oroville plant of the Stewart Calvert Company, for treatment, where the crude salts were dissolved and recrystallized for sale to the drug and tanning industries.

Basque Chemical Co.

The Basque Chemical Company is operating a series of five lakes situated fifteen miles west of Ashcroft, B.C. From these lakes they are excavating crude magnesium sulphate (Epsom salt), which is shipped to Vancouver, where it is refined and sold as medicinal salts and to the tanning industry. Plans are now under way for greatly extending the development of this deposit.

Clinton, B.C.

There is a small lake one mile south of the town of Clinton, B.C., from which a small tonnage of magnesium sulphate was obtained a few years ago. This deposit was not operated during 1922.

Lillooet Soda Company

In the vicinity of Meadow lake, fifty miles north of Clinton, B.C., the Lillooet Soda Company is operating a sodium carbonate lake and shipping the product to Vancouver over the Pacific Great Eastern railway. This material is being disposed of at the present time to the soap manufacturers of the Pacific coast, but plans are under way to dry the material at the lake and increase the output.

This company also owns several other lakes of the same material which they propose to operate at some future date.

INDUSTRIAL USES OF GLAUBER'S SALT AND EPSOM SALT

In the chemical manufacturies, Glauber's salt and Epsom salt find a very extensive application, and the demand for these products is ever increasing. Since by far the greater part of the Canadian consumption is imported, it is of importance to analyse their different uses with a view to finding whether material from Canadian deposits cannot be utilized.

Sodium sulphate is used extensively in the pulp and paper, glass, dye and textile industries, and to a smaller extent for medicinal and tanning purposes. Magnesium sulphate is employed for tanning and dyeing, and for textile and medicinal use.

Pulp and Paper Industry

The manufacturers of kraft paper are very large consumers of salt cake, using annually over 50,000 tons. The principle upon which this process depends is the solvent power of caustic alkali on the non-cellulose constituents in the wood. Consequently sodium sulphate cannot be used directly, but has to be converted into caustic alkali and dissolved in water. This solution is termed white liquor and is composed mainly of sodium hydrate and sodium sulphide. To prepare this white liquor, the so-called black liquor, containing the dissolved non-cellulose substances, obtained from the process, is evaporated to 35° Be and put through a rotary furnace. At the discharge end of this furnace, enough sodium sulphate is added to replace the alkali lost in the treatment of the wood. It is then shovelled into smelting furnaces where the sodium sulphate is reduced to sodium sulphide and carbonate. From the smelting furnace the melt flows in a molten condition to dissolving tanks containing water or dilute washings from the sludge in the causticizing room. When the solution has reached the desired density it is discharged into the causticizing system and lime is added. The reaction with the lime is as follows:—

$$Na_2CO_3 + CaO + H_2O = 2NaOH + CaCO_3.$$

Magnesium sulphate is used in the paper industry for weighting paper.

Glass Industry

Manufacturers of glass bottles and other containers use comparatively small amounts of sodium sulphate or salt cake (Na_2SO_4), according to some manufacturers only about 40 lbs. per ton. Since it gives a more homogeneous batch than when soda ash is used, a very clear glass is obtained, therefore it can be advantageously used in window glass and where very clear and white glass is required. The window glass manufacturers require about 6-10 per cent sodium sulphate (Na_2SO_4) but foreign manufacturers often add as high as 25 per cent to their batch.

Tanning Industry

Sodium sulphate is not used in the tanning industry proper, but comparatively large amounts are used advantageously in curing hides, superseding salt. It is difficult to arrive at a figure of the requirements for that purpose. However, large amounts of sodium sulphide are used mixed with lime, or by itself. In practice, sodium sulphide is always obtained by the reduction of sodium sulphate. The immersion of hides in a solution of sodium sulphide removes the hair quickly, freshens the grain, and imparts great toughness to the hide. A tannery using sodium sulphide will require as high as 75 tons per year for an output of 180,000 hides.

Magnesium sulphate is used by the tanneries in manufacturing sole leather, the estimated annual requirement for Canada being about 1,000 tons. It is used in sole leather only, to get a clean, shiny cut; it also helps to retain the moisture in leather and increases its weight.

Dye Industry

The present estimated annual consumption of Glauber's salt by the dye works is from 1,300 to 1,500 tons. The dyers prefer the natural Glauber's salt to manufactured salt cake as it is free from nitrates and nitrites which readily oxidize the dye. Magnesium chloride tends to disassociate the colours.

The effect of Glauber's salt under the general theory of dyeing is, mechanically retarding the interaction between the colour-acid and the fibre, chemically retarding the liberation of the colour-acid from the dye salt, and in affecting the solubility of the dye stuff solution.

The general effect of the addition of sodium sulphate appears to be a more uniform distribution of the dye, since this substance has a slight solvent action on the dye taken up by the fibre, thus removing it from those places in which the dye may have been deposited in excess and generally retarding the operation of dyeing, thus obtaining a slow and even setting.

Epsom salt is also used in the dyeing industry but not to a very large extent. In some cases it is used in the after treatment in order to increase the fastness to washing. Sodium sulphide is extensively employed for the sulphur dyes.

Medicinal Use

For medical purposes Glauber's salt is employed as an aperient and is one of the safest and most innocent known. It is to some extent used in cattle food and especially for veterinary uses. Also, it has been employed in cooling mixtures. Magnesium sulphate has an extensive use in medicine and very large quantities are used for this purpose, estimated at about 500 tons per annum.

Textile Industry

Magnesium sulphate is extensively used in textile manufacture for various purposes. In bleaching wool where sodium peroxide is used, since caustic alkali attacks the wool, magnesium sulphate is added to destroy its corrosive effect. It is also used for weighting textile fabrics, especially silk. Printed cotton is always "finished," the finishing material consisting of a mixture of kaolin, gypsum, etc., and magnesium sulphate. Magnesium sulphate mixed with gypsum and ammonium sulphate is used for the manufacture of non-inflammable fabrics.

III

VOLCANIC ASH NEAR WALDECK, SASK.

Location of Deposit

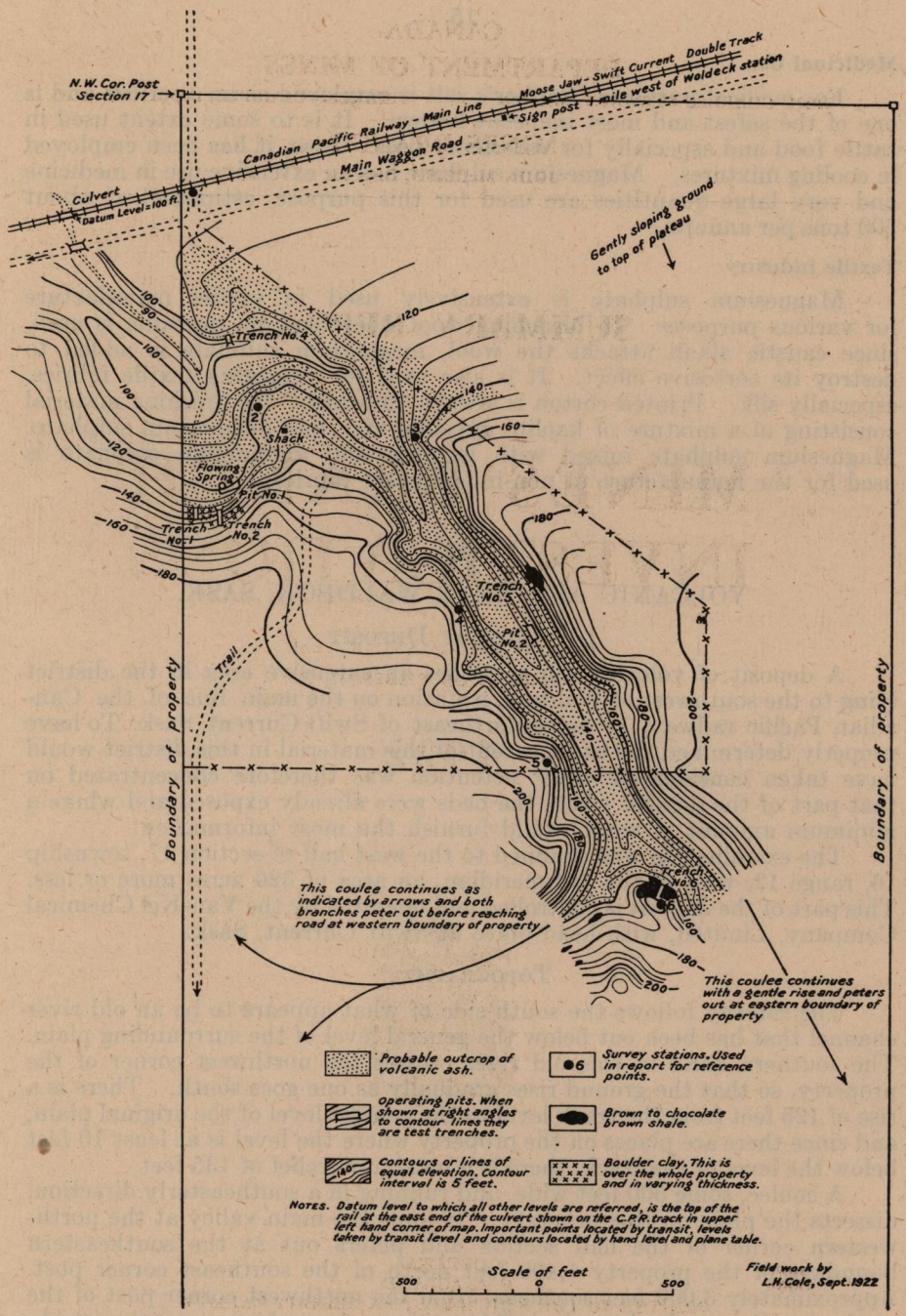
A deposit of volcanic ash underlies an extensive area in the district lying to the southwest of Waldeck, a station on the main line of the Canadian Pacific railway, 11 miles northeast of Swift Current, Sask. To have properly determined the exact extent of this material in this district would have taken considerable time; attention was therefore concentrated on that part of the deposit where the beds were already exposed and where a minimum amount of work would furnish the most information.

The examination was confined to the west half of section 17, township 16, range 12, west of the 3rd meridian, an area of 320 acres more or less. This part of the deposit is controlled and operated by the Van-Kel Chemical Company, Limited, with head office at Swift Current, Sask.

Topography

The railway follows the south side of what appears to be an old river channel that has been cut below the general level of the surrounding plain. The southern bank of this old river crosses the northwest corner of the property, so that the ground rises gradually as one goes south. There is a rise of 125 feet from the track level to the general level of the original plain, and since there are places on the property where the level is at least 10 feet below the level of the track, there is a maximum relief of 135 feet.

A coulee, some 500 feet wide, and running in a southeasterly direction, dissects the plain on this property. It joins the main valley at the northwestern corner of the half section and peters out at the southeastern boundary of the property 1,400 feet north of the southeast corner post. Approximately 3,000 feet southeast from the northwest corner post of the section another coulee branches off from the main coulee in a southwesterly direction and peters out near the western boundary.



Map of the property of the Van-Kel Chemical Co., Ltd
W½ Sec. 17, Tp. 16, Range 12, west of the 3rd meridian
Saskatchewan

FIG. 1.

With the exception of these two coulees and that portion of the old river channel already mentioned, the surface of the property is gently undulating, with a maximum relief of less than 20 feet. The main topographic features are shown on the accompanying map. (Fig. 1.)

The presence of these coulees and the old river valley have rendered the beds underlying the general mantle of boulder clay or till readily accessible, since they have cut through the drift material and portions of the underlying shale beds, leaving in many places only a thin veneer of covering on top of the ash beds.

Geology

It would be necessary to examine a considerable area of the surrounding country to determine whether the beds noted were continuous or only local, and whether the dips and strikes held good over the whole deposit. The time available did not permit of such detailed work, so that the data obtained can only be considered as applying to that part of the deposit examined.

The geological sequence as noted on this property was as follows:—

Boulder clay or till	Varying depths
Brownish grey shale	Thickness undetermined, 20 feet uncovered in one trench
Volcanic ash beds	Thickness of 37 feet in one place
Blue-grey shale?	Undetermined

Boulder Clay or Till

The boulder clay or till is the common type to be seen in the territory lying to the east of Swift Current. It consists of a heavy clay matrix in which numerous pebbles and boulders are embedded. Many of these boulders are badly disintegrated and readily crumble in the hand when pressure is put on them. This is especially true of many of the granite-gneiss pebbles contained in the clay. There are a number of remarkably well-rounded hard quartzite pebbles embedded in the boulder clay, especially at a definite horizon in the drift. The larger number of the pebbles and boulders are of Archean origin. The boulder clay is of varying thickness in different parts of the property. For example, trench No. 2 was cut in boulder clay horizontally into the hill for a distance of 14 feet and then sunk for a depth of 5 feet below this level, so that at the end of this trench there was a face of boulder clay of approximately 13 feet thick. In this trench, as well as in trench No. 1, the boulder clay was found resting directly on the ash beds and the brown shale was not encountered. It could not be definitely determined whether the top of the ash beds in these two trenches was the true top or whether some of the material had been eroded prior to the deposition of the boulder clay. At trenches Nos. 5 and 6 the overburden of boulder clay was less than one foot in thickness.

The boulder clay is very hard and compact and it would be difficult and expensive to do any extensive stripping where this material is three feet or more in thickness.

Brownish Grey Shale

In trench No. 6, beds of brownish grey shale were uncovered and a thickness of 20 feet exposed. The shales exposed in this trench have been highly weathered and carry some iron oxide, which gives it a chocolate brown colour when first exposed, but which afterwards turns to a brownish grey. These shales are highly arenaceous or sandy in texture and will probably prove suitable for a fair class of building brick, judging from preliminary tests made by Professor W. C. Worcester of the University of Saskatchewan. It is quite probable that this shale extends over a larger area and to a higher level than that exposed in the top of trench No. 6, since the material thrown out of badger and gopher holes shows similar shale all the way up the southwestern coulee nearly to the level of the general plain.

The contact between this shale and the underlying ash beds was encountered in trench No. 5. In this trench the materials gradually grade into one another so that it is difficult to determine the exact point at which the change occurs.

Volcanic Ash Beds

It is the material from these beds that is used in the manufacture of domestic cleansers and hand cleaners. The ash is a loosely compacted, finely divided material, and will vary, when dry, from a light buff colour to a pure white. The strike of the beds could not be determined with any degree of certainty, but it is approximately northeast, with a dip to the northwest of less than 10 degrees.

Two pits have been opened on the property and material has been used from both of them. Pit No. 1 was opened at the foot of the rise near the western boundary of the property and exposed very white material of fine texture. A strong flowing spring was encountered at this point which prevented the pit being operated to any depth without draining. A test hole, however, was sunk in the bottom of this pit, with an auger, to a depth of 10 feet and was still in the ash beds. The elevation of the bottom of this test hole was 110 feet,* and since in trench No. 2 the ash beds were encountered at an elevation of 131 feet, there is a thickness at this point of at least 21 feet.

In pit No. 2 the elevation of the bottom is approximately 130 feet. A test hole sunk in the bottom of this pit to a depth of 10 feet encountered the base of the ash beds. Trench No. 5 gave the contact of the ash beds and the brown shale at an elevation of approximately 157 feet, so that in this part of the deposit there is a thickness of 37 feet. The ash was not encountered in trench No. 6, but 50 feet north of this trench a shallow hole uncovered it at a few feet lower than the bottom of the trench. Trench No. 4 did not show the ash beds but exposed what appeared to be a bluish grey shale, but this could not be determined definitely. It was very similar in appearance to the material found in the bottom of the test hole sunk in the bottom of pit No. 2. Numerous badger and gopher holes also indicate the character of the underlying material.

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*Datum level C.P.R. track at east end of culvert = 100 feet.

It is reasonably certain that the ash beds are continuous between the two pits and also over a considerable area in the district surrounding the property, since these beds are found at the foot of the old shore line one mile southwest of the property, and also in the bottom of a well at a depth of 20 feet on the next section west. Four miles directly south of the property material very similar in appearance is found, but the analysis shows that the material from the more southern locality carries a considerable amount of gypsum, which is not present in the deposit in question.

Material Available.—Assuming that the beds are continuous over the whole of the property, except on the area to the northwest where the ash is known to be absent, and assuming that the thickness holds nearly constant, there is a large tonnage of the volcanic ash present. The larger part is covered with an overburden of boulder clay and shale. There is, however, a large area with only a light covering, and this material will be sufficient to supply the company for many years. It is impossible to make an accurate determination of the tonnage available without further work in proving up the thickness of the material and the extent of the overburden, but taking the outcrop of the ash beds as shown on the map to be correct, and if the material were excavated to its full depth for an average width of 250 feet, there would be available approximately 200,000 tons in a strip 500 feet each way from pit No. 2 in the large coulee. From this it can readily be seen that by confining work to the outcrop of the ash beds alone, there is a large tonnage available with only a small amount of stripping.

Composition.—There has been a diversity of opinion regarding the composition of this material. Diatoms were reported by Andrews and Cruickshank, of Regina, Sask., in a sample they examined from pit No. 1, but in all the samples taken by the writer no diatoms were noted under the microscope. Five samples were submitted to Eugene Poitevin of the Mineralogical Division of the Geological Survey who reported on them as follows:—

> The enclosed five specimens sent in for examination consist of semi-indurated volcanic ash composed essentially of glass particles with only minor proportions of comminuted crystals and microlites.

It is quite conceivable that where the spring comes out in pit No. 1, diatoms may be in process of deposition at the present time and therefore samples taken from this locality might easily contain occasional diatoms, but for all practical purposes the material can be considered volcanic ash.

The composition of the samples is shown by the following analyses:—

	1	2	3	4	5	6	7
SiO_2	64·82	65·18	69·76	41·28	85·04	91·08	86·10
$Fe_2O_3Al_2O_3$	21·64	19·70	15·56	9·10	4·04	0·96	4·90
CaO	2·25	2·15	2·20	17·25	0·67	0·36	1·30
MgO	2·77	1·99	0·90	4·18	0·66	0·29	0·80
SO_3	.....	.....	.....	10·34	.....	.....	.....
Loss on ignition	8·52	10·98	11·58	17·85	8·05	5·98	6·50
Totals	100·00	100·00	100·00	100·00	98·46	98·67	99·60

Localities from which samples were obtained:—

1. Buff material, Pit No. 2.
2. White material, Pit No. 2.
3. White material, Pit No. 1.
4. Outcrop of white material 4 miles south of Van-Kel property.
5. Analysis by Frank T. Shutt of material from same locality as sample No. 2.
6. Analysis by Frank T. Shutt of material from same locality as sample No. 3.
7. Analysis by Andrews and Cruickshank, Regina, Sask., of material from Van-Kel property, exact locality not stated.

Notes on Analyses.—First four samples were analysed by F. MacNiven, Mines Branch. Loss on ignition in first four samples obtained by difference; in sample No. 7 it includes moisture. In samples Nos. 5 and 6 amount under SiO_2 is "rock matter insoluble in acid." It is probable that in samples Nos. 5, 6, and 7, the fusion for silica was not made, hence the greater part of the iron and alumina is included in the silica figures, thus accounting for the high results.

An analysis of the water from the spring in pit No. 1 gave the following results:—

	Parts per million.
$CaSO_4$	290·0
$MgSO_4$	175·0
NaCl	27·5
Na_2SO_4	2,870·0

The ash is very fine in texture and has a high power of absorption. When first quarried it breaks up into powder very readily in the hand, but becomes slightly harder on exposure to the air. It does not appear to have a very strong bond. This quality of readily absorbing liquids, along with its abrading power, makes it a good material for making domestic cleansers. When burned at a comparatively low temperature, such as is employed when making the Little Wonder Fire Lighter bricks, the material still retains its high power of absorption, and it is this fact which is made use of in the application of the Fire Lighter.

The operating company hauls the material by wagon to Swift Current where they operate a small plant and turn out a line of cleansing products which seem to be finding a ready market in the prairie provinces. They have placed on the market the following lines.

Trade names:—

Busy Bee Cleanser
Busy Bee Hand Cleaner
Busy Bee Odorizer
Little Wonder Fire Lighter

All of these products have as a base the volcanic ash obtained from the company's property.

Very little has been done to determine the extent to which this material is suitable for commercial purposes, and it is possible that many other uses could be found for it besides those in which it has already been employed. The manufacture of bath brick, insulating brick, varying grades of metal polishes, and many other products are worthy of consideration.

IV

CANADIAN FELDSPAR IN 1922

V. L. Eardley-Wilmot

Although the shipments of feldspar from the Dominion have been slightly less than in 1921, there has nevertheless been considerable activity in the industry, which appears to have gained further impetus during the latter part of the year.

In Table I will be found the shipments of feldspar by provinces during the last ten years.

Practically the whole of this material has been shipped crude to the United States grinding mills. Less than one per cent of the 1922 shipments were ground in the local mills at Kingston and Toronto. The products of these latter were used by the grinders or sold to Canadian manufacturers. The recent duty of 30 per cent on ground spar imported into the United States has doubtless been the cause of such a small amount of local ground material, but a probe into the European markets may reveal an outlet for the higher priced form of the product.

The average Canadian feldspar appears to be of better quality than that of the United States, which latter is mainly in the form of graphic granite.

Canadian feldspar is in much demand by various manufacturers in the ceramic industry in the United States for high grade pottery and porcelain purposes, for enamels and glazes, and in the electrical industry for insulators. The lower grade material is used in building materials such as stucco work and roofing, also for scouring soaps, opalescent glass, poultry grit, etc., etc.

In England feldspar is mainly used for enamelling of metal ware, the consumption, according to the Imperial Institute Mineral Resources Commission, being in the neighbourhood of 5,000 tons a year. The main source of supply is Scandinavia, the feldspar being delivered at an English port for about £2-15 per long ton. In the potteries china clay, china stone and mica clay, brought from Cornwall (England) are chiefly used.

In Table II will be found a general list of the feldspar grinders.

Many feldspar producers do not fully appreciate the difficulties of the manufacturers of pottery and porcelain. Certain formulae are calculated for the admixture of ingredients, and the temperature at which fusion has to take place (by the Orton cone) is carefully worked out for the particular grade of spar which they are using. If a batch of different grade of material is mixed in, it is liable to ruin the whole kiln run by either altering the fusion point or by contamination from discoloration and specking. A manufacturer cannot be continually altering his formulae and standard adjustments, and naturally prefers to buy from an experienced producer who has a large deposit of a regular grade. It is perhaps for this reason that the producer, or prospector, who has just begun feldspar mining often finds a difficulty in disposing of his product at a No. 1 price. It might pay him to develop his property and sell his first few carloads at a low figure to a grinder, or local buyer, and thus prove that he has a reasonably large deposit of good grade spar, or otherwise. When satisfied

that his property meets all requirements he should have no difficulty in getting a representative of one of the large grinding plants or manufacturers to come and see the deposit and make a contract for his output.

There is no doubt, however, that some kind of standard or guide is required to enable the producer to correctly grade his material, and it is hoped that some practical solution will be evolved. Although there has lately been a considerable amount of discussion devoted to this subject, which was also brought up before the recent ceramic meeting in Pittsburgh, so far no definite standard has been universally adopted.

QUEBEC

Buckingham District

During the year feldspar mining has been very active throughout the Buckingham district, from which the whole output for Quebec has been obtained. About a dozen deposits were worked by different individuals or companies, and approximately 12,500 tons of spar were shipped, about three-fifths of which came from the Derry mine owned by the O'Brien and Fowler Company, Ltd. Mahoney-Rich and the Pednaud quarries together shipped a little under 5,000 tons. A few hundred tons of dental spar and No. 2 material was shipped by Bush Winning, and the remainder came from five small producers.

The majority of the spar shipped from this district is very high grade and is of an almost pure white or buff colour. The deposits to the north of Buckingham occur as dikes, containing exceedingly large and pure crystals of feldspar and quartz which can easily be separated and sold as separate products. In the deposits worked by the principal producers the injurious ingredients such as mica, pyrites, hornblende and tourmaline are not usually intimately mixed, but when they occur are more or less concentrated at the edges of the dikes. Buff screenings from the Derry quarry are crushed and used for stucco work for ornamental building purposes. Practically all the deposits that are in the vicinity of Glen Almond, 8 miles north of Buckingham, occur as dikes carrying massive white or buff crystals of spar and of bull quartz, while farther south the spar appears to be of a pinker colour, not so massive, and carrying the quartz and impurities more intimately mixed.

The Derry mine on range I, lots 7 and 8 of Derry tp., operated by N. B. Davis for O'Brien and Fowler, has during the year shipped 50 per cent more spar than any other individual producer in Canada. It was originally staked by A. Parcher and R. J. Donaldson in 1917, and consists of a feldspathic dike about 100 feet wide with a well defined west wall of gneiss. This dike has now been quarried and stripped over a distance of about 400 feet along its strike. Operations were started in August, 1920, and the workings are at present confined to a level entry quarry on the northeast side of a gulley. This quarry is some 300 feet long, 75 feet wide, and 40 feet deep at the north face, and is worked in bench fashion, and recently a bench has been started from below the track level. Crystals of pure spar 30 feet long and 15 feet wide are said to have been encountered.

Up-to-date methods are used for sorting and extracting the material, which is trammed out of the quarry over a high trestle alongside the road. The No. I spar is dumped into a loading bin, and the quartz, fines, and

waste material are dumped separately. The marketable materials are conveyed by wagon along the company made road to their own landing stage at Glen Almond on the Lievre river, from which they are loaded into 40-ton scows by a special dumping device, and conveyed by local steamer down to the railway siding at Buckingham. The quartz is sent to the Buckingham Chemical and Reduction works; the fines to the Company's crushing plant at Ottawa where it is further ground and sized for stucco and other purposes. In winter the material is hauled to Buckingham, 9 miles by snow road.

The No. 1 feldspar is shipped crude to grinding mills in the United States. The ground material is in high demand for the pottery trade and particularly for the manufacture of high tension insulators. It is stated that the quartz content of the Derry spar recently mined is only 1·5 per cent, with practically no sorting. An average analysis shows 13·0 per cent potash and 1·7 per cent soda.[1]

The O'Brien and Fowler Co. also produce a clear semi-transparent spar from the centre of their deposit, which is sold for dental purposes under the trade name of Glen Almond spar.

About one mile farther north, on range II, lot 9, Mahoney and Rich of Ottawa, during the latter part of the year, took over the contract of Higginson and Parker and have been operating a property on top of a steep hill. The grade and quality of the dike are somewhat similar to the Derry mine, though the deposit does not appear to be as extensive, and mining operations have to be confined to open pit sinking and derrick hoisting. A pit about 85 feet long and 60 feet deep has been sunk on the dike. Steam power operates the derrick hoist and drills, and during the summer a 30-ton Holt caterpillar tractor hauled the material to a landing stage on the Lievre river, whence it was conveyed by scow to the Buckingham siding. Some very pure grade spar and quartz have been extracted from this deposit, which is the second largest producer in the province, approximately 3,500 tons having been shipped from the pit during the year. At the end of the year, however, owing mainly to trouble with water, Mahoney and Rich ceased operations and have moved their outfit to the McGivern property in Buckingham, range X, lot 5, about 8 miles to the southeast, which is owned by the New York Feldspar Corporation and upon which they have begun their 1923 operations.

One mile south of the Derry mine in Buckingham, range XII, lot 14, G. Pednaud operated his quarry throughout the greater part of the year. The workings consist of a series of level entry quarries one above the other over a vertical height of 300 feet. The deposit was originally worked for quartz, but the occurrence of large clean crystals of feldspar, particularly on the topmost workings, led to appreciable shipments of this material as well as quartz. Practically the whole spar output was taken by the O'Brien and Fowler Company.

In this area of about 4 square miles, which includes the above mentioned deposits, Messrs. Parcher, Whitmore and Louis Couture, have done some prospecting work on different deposits and have laid aside a few tons of spar and quartz which they intend to haul out by sleigh during the winter months. A little farther south J. J. Cameron sunk a pit on a dike

[1]See description Can. Min. Inst. Bull. 118, February, 1922, pp. 229-35. By N. B. Davis.

on the Gorman property, but abandoned the work after extracting about a car load of material. Operations have recently been begun on another part of the same dike by T. F. Foran.

Towards the end of the year A. J. McMillan did some prospecting on a large dike on his farm that has not been worked for over 20 years and on which there is evidence of local patches of fair grade pink spar. Practically all the output from the smaller producers has been taken by O'Brien and Fowler.

On Portland E., range IX, lot 3, Bush Winning has been operating the Villeneuve Extension quarry. This property is about one and a half miles due south of the well known Villeneuve mine, which years ago yielded dental spar and some of the highest grade spar ever mined in the Dominion. The appearance of the material on the Winning property is in many respects very similar and occurs in massive crystals of pure white spar and bull quartz. The quarry workings are on the side of a steep ridge, from which during the year several car loads of dental spar and No. 2 spar have been shipped. If it were not for the long haulage distances a considerably greater quantity of spar would be shipped from these more northerly deposits. In Table III will be found the dental spar shipments since 1910.

The majority of the feldspar operators in this district have shipped quartz to the chemical works at Buckingham.

St. Lawrence Feldspar Company, Quetachu Bay, North Shore of Gulf of St. Lawrence, Quebec

The feldspar found at Quetachu bay, opposite the centre of Anticosti island, occurs as a large batholith of pegmatite which covers an area of several square miles and which has irregularly intruded into black hornblende-biotite-gneiss, leaving many tongues and islands of the latter rock within its mass.

Generally speaking surface indications point to an intimate mixture with black mica and garnets, with smaller amounts of tourmaline, white mica and alteration products of all these minerals. The whole mass seems to be intermixed with smoky and white quartz, present either in graphic intergrowth with the feldspar or segregated as irregular nodules.

From an examination of the outcrops it would appear that the most valuable portion of this great batholith occurs within the 200 acres comprising the St. Lawrence Feldspar Company's property.

Recent attention has been turned to a promising looking area, which will be referred to later, but an account of the previous operations and conclusions thereby arrived at are as follows:—

Up to the end of 1918 prospecting work was confined to the southwestern end of the peninsula upon which this property is situated. Several hundred tons of material, which would be classed as Grade 3 spar, have been stock piled. A few tons of No. 1 spar have been cobbed out from the oldest stock piles, and a few more tons of the same material could be sorted out from the rock excavated in 1918. It is understood, however, that this excavated rock is now being used in the construction of the new wharf and piers. There is of course a possibility of finding a market for the lower grade material, such as for scouring soaps, provided that the excess of the intermixed quartz and garnets can be economically cobbed out.

As a result of this prospecting work and subsequent examination, which showed evidence of only small and scattered patches of higher grade material, it seems doubtful whether a good body of commercial spar for pottery purposes exists, either in this portion of the peninsula or in the islands. (An accurate description as applied to this area will be found in the report of H. S. Spence, "Feldspar in Canada," issued by the Mines Branch in 1916).

Consequently the idea of further prospecting for commercial pottery spar in this area has been temporarily abandoned, and attention has recently been turned towards the northeast, where appearances improve and the harmful impurities are not so intimately mixed. Several local belts of clean high grade spar, and indications of a large area of good graphic granite, were noticed here on the surface, but practically no prospecting work has as yet been done to prove these more promising looking showings.

The Company is building a wharf and has laid down a tram line connecting it with the face of a steep bluff, on the top of which the last mentioned showings of commercial spar are exposed. It is understood that the Company propose to develop this body in the near future. By cutting into this bluff a good working face will be exposed, by means of which such commercial spar as may be opened up can be economically quarried and then sorted and trammed to the wharf. It is proposed to devote the winter's operations to constructional work, further development of the new showings and possibly a little prospecting.

All the No. 1 spar and graphic granite mined will have to be carefully hand sorted, and with efficient cobbing it should be possible to obtain a considerable tonnage of good material.

Some of the purer patches contain individual crystals of as high grade a spar as the writer has seen in the Dominion.

In many places deep water extends to the deposit, with good anchorages which are said to be well protected from storms, so that the loading of the mineral into ships should present no difficulties.

The feldspar output for Quebec in 1922 constitutes a record for the province, being about 3,000 tons higher than the previous record in 1921. It is of interest to note that for the first time the value of the Quebec shipments has slightly exceeded that of Ontario though the actual tonnage is less. Feldspar was, however, shipped from Quebec ten or twelve years previous to Ontario, which commenced in 1900. Most of those old Quebec shipments, which reached a maximum of 3,000 tons in 1899, came from Templeton township and were valued at $2.50 per ton, which was considered high compared to the early Ontario shipments at $1.25 per ton.

ONTARIO

There has been a slight falling off in the Ontario feldspar shipments for the last two years when the maximum shipments of 37,000 tons was reached in 1920. This is mainly due to the inactivities of the larger Verona quarries, including the Richardson quarry, which for a period of 15 years, had an average annual output of over 10,000 tons. It is stated, however, that there are now thousands of tons of commercial spar that have been picked over from the dumps and are ready for shipment.

During the year the greatest production came from the Perth district.

Perth Area

In southwest Lanark feldspar mining has been active throughout the year. A series of dikes, having a general northeast-southwest strike, yielding commercial spar have been mined in Bathurst, con. IX, lots 20 and 21, about 8 miles northwest of Perth.

Practically the whole output of some 6,000 tons came from this small area, over three-fourths of which was shipped by the Rock Products Company, the second largest producer in Canada during the year.

The workings of this Company, which are alongside the road, consist of a pit about 75 feet long by 45 feet wide and 55 feet deep, and during the latter part of the year the Company drove a tunnel from the bottom of the northeast face. The best spar is of a very dark pink colour, and although the run-of-mine ore carries appreciable impurities, the deposit does not appear to be extensive and contains local patches of soda spar; nevertheless by means of a practical system of cobbing by experienced men, an excellent grade of No. 1 spar is separated and shipped to the Company's grinding plant at Toledo, Ohio. The mine is equipped with up-to-date hoisting machinery, including a substantial lattice steel derrick.

About a quarter of a mile to the east S. H. Orser has been working a deposit of spar from which he shipped a few car lots. The material is, however, somewhat low grade and requires considerable hand cobbing, so that work was abandoned during the late fall.

To the west of the Rock Products, and on the same lot, A. M. Campbell has taken an option on a feldspathic dike of considerable size. The outcrop, which is on top of a ridge, is some 800 feet long and 300 feet across, in which occur a few lenses of gneiss. Surface showings indicate a certain amount of impurities and admixed quartz, but recent work at the east end of the body has shown improvements. The dike is of sufficient size to allow of there being several local workings, any one of which would constitute a small shipper in itself. The spar is of the usual pink variety, but a dike of almost white spar, not unlike the north Buckingham mineral, has been exposed. Work was started during the summer, and a derrick operated by horse and drum was erected, but this has now been duplicated. Almost 1,000 tons are believed to have been shipped which have been classed as No. 2 grade, but as a result of the most recent operations some of the surface stained material has been cleaned off and the owner expects to be shipping No. 1 spar during 1923.

During December a few cars of spar were shipped by S. H. Orser from one of R. McConnell's properties known as the Kirkham quarry. The dike appears to carry a fairly large proportion of impurities so that considerable cobbing is necessary to obtain clean spar. A small crushing plant was erected by Orser on the Burns property, where the railway cut has exposed a dike north of Christie lake, and a few car lots of crushed quartz and stucco material were shipped during the summer.

Prospecting work has been done on two or three other deposits in the district but no shipments have yet been made.

Bancroft Area

Mining and prospecting have been conducted during the year on about half a dozen spar properties in Monteagle township in the vicinity of Hybla station, some 8 miles north of Bancroft, and approximately 5,000 tons of spar have been shipped. About half this output came from the quarry of the Feldspar Mines Corporation, a subsidiary of the Pennsylvania Pulverizing Company.

The workings consist of a quarry some 200 feet long by 60 feet wide and 20 feet deep at the southwest face. This dike, although containing several belts of impurities and admixed quartz masses, yields a considerable quantity of clean spar, particularly at the southwest face, where the present workings are conducted. Several patches of pure green spar, known as amazonstone, have been mined, a carload of which has been shipped. The Company has an up-to-date steam hoisting and drilling plant.

On the adjoining lot the Verona Mining Company, which was a subsidiary of the Pennsylvania Feldspar Company, made considerable shipments of spar from the McDonald property, leased to them by W. A. Hawley of Toronto in 1919, since when about 15,000 tons have been shipped. The Company has now gone into liquidation, and the property was taken over at the end of the year by the Genesee Feldspar Company, who have re-started operations and have erected derricks and installed machinery. The spar body, which is on the top of a hill, appears to be of large extent, and although locally carrying the usual impurities and aggregates of smoky quartz, nevertheless contains large crystals of pure spar. The level entry quarry is some 400 feet long and in places 80 feet wide, but in the wider portions the country rock of black gneiss, which caps over part of the dike, has been considerably broken into.

A short distance from the above the Monteagle Feldspar Company operated the Watson property, but during the summer, after shipping a few car lots, the Company went out of business and was taken over by the Cleveland Feldspar and Products Company.

The spar body consists of a narrow dike running into the side of a valley. From the face of a small quarry a tunnel, some 120 feet in length has been driven, from which the spar was trammed out and dumped into a loading bin above the roadway that runs along the valley. A few prospect pits have been opened up on the top of the hill, exposing a fair grade of spar, which, however, would require careful sorting.

In all the above deposits crystals of the radium bearing mineral euxenite were found, particularly in the last mentioned dike.

Work has recently been started on a likely looking spar body on the farm of J. Thompson, a little to the north of the Feldspar Mines Corporation property.

Verona District

There are a large number of feldspar quarries in southern Frontenac county, particularly in the Verona district, from which at one time practically the whole Canadian output of the mineral was obtained. The Feldspar Quarries, Ltd., a subsidiary of the Dominion Feldspar Corporation, and the Gardner property, leased during the latter part of the year to W. G.

Treadwell, were the only two producers of any importance, both of which are situated in the north of Loughborough township. Approximately 3,700 tons of spar were shipped from the district during the year. A small portion of this consisted of material sorted out from old dumps. During the summer the former Company leased the property from W. A. Dillon and T. Craig who had previously shipped several hundred tons of spar.

Although the Feldspars, Limited, have shipped practically no ore from any of their properties, including the well known Richardson mine, nevertheless parts of the huge dumps have been sorted over, and some six or seven thousand tons are stated to be ready for shipment as soon as the railway transportation facilities, now under way, are completed. Preparations are also being made to sink on the extensions of the Richardson lens.

During the summer W. Gardner operated a small crushing and screening plant on his property and has shipped, besides crude spar, several hundred tons of stucco and chicken grit material.

The Frontenac Floor and Wall Tile Company, whose plant is at Kingston, has recently installed a feldspar grinding unit of about 1,200 tons a year capacity. The majority of the material that they grind, all of which comes from the Verona district, is used in the manufacture of their own tiles, which have proved very satisfactory and durable.

The Feldspar Milling Company have also ground some of the Verona spar as well as some from Buckingham. The Company's mill, which is situated in Toronto, consists of a chaser mill, screens and two 8 ft. x 6 ft. pebble mills. The power is supplied by electric motors.

Other Localities

The Canadian Non-Metallic Minerals, Ltd., has been operating a property at Aylen Lake, near Opeongo in Dickens tp., con. III, lot 12.

The work done consists of a level entry quarry leading from the railway track. The cutting is some 100 feet long and 40 feet wide with a 20-ft. high face. A spur track runs into the quarry, which enables direct loading into the cars. Several hundred tons of a dark pink spar have recently been shipped.

About two miles to the east another feldspar dike was operated by the company during 1921 and upon which machinery consisting of a compressor, hoist, boiler and dynamo, etc., is installed.

The Industrials Minerals Corporation, whose president is H. E. Harcourt of Toronto, have been shipping feldspar from quarries in Monmouth tp., and also in Dryden tp., in the Sudbury district. The former is of low grade, consisting almost entirely of graphic granite. The latter, leased from the McPhee Brothers, is in con. II, lot 9, near Coniston. The first shipment, which was made by the Corporation at the end of the year, consisted of high grade flesh pink spar, showing very few impurities.

The Wheeling Feldspar Company did a little work in their quarry on the north shore of Cecebe lake in Parry Sound county, Chapman tp., con. II, lot 26. This property was operated two years ago and the spar is now treated by means of a grinding and screening plant and then sold in the United States as stucco. The crushed material is shipped down the lake in a scow, thence to Burks Falls station, 9 miles to the east.

TABLE I

Canadian Feldspar Shipments During Last Ten Years [1]

Year	Quebec			Ontario			Dominion		
	Tons shipped	Value	Average price per ton F.O.B. cars	Tons shipped	Value	Average price per ton F.O.B. cars	Tons shipped	Value	Average price per ton F.O.B. cars
		$	$		$	$		$	$
1913.....	74	1,554	21 00[2]	16,716	59,241	3 54	16,790	60,795	3 62
1914.....	98	2,156	22 50	17,962	68,668	3 82	18,060	70,824	3 92
1915.....	572	2,005	3 50	13,987	55,796	3 98	14,559	57,801	3 97
1916.....	4,610	18,075	3 82	14,878	53,332	3 57	19,488	71,407	3 66
1917.....	1,188	8,204	6 87	18,274	81,622	4 47	19,462	89,826	4 62
1918.....	191	4,279	22 40[2]	18,591	108,449	5 80	18,782	112,728	6 06
1919.....	925	13,073	14 15[2]	13,754	73,158	5 32	14,679	86,231	5 85
1920.....	649	10,052	15 50[2]	37,224	270,843	7 28	37,873	280,895	7 45
1921.....	9,737	80,180	8 58	20,115	150,457	7 43	29,868	230,754[3]	7 73
1922[4].....	12,672	114,235	9 00	15,800	113,850	7 20	28,472	228,085	8 00
Total....	30,698	253,013	.....	187,301	1,035,416	.....	217,952	1,289,346	.....

[1] From the "Mineral Production of Canada."
[2] High price due to shipments of dental spar.
[3] Includes 16 tons valued at $117 from Nova Scotia.
[4] Approximate; final statistics not yet published.

TABLE II

Feldspar Grinding Plants in Canada and in United States

Canada

Name of Company.

Feldspar Milling Co., 33 Richmond St. W., Toronto, Ont.

Frontenac Floor and Wall Tile Co., Kingston, Ont.

United States

Bedford Mining Co., Bedford, N.Y. Operating two mills at Bedford, N.Y., and grinding potash and soda feldspar from Westchester county, N.Y., exclusively.

Carolina Feldspar Co., Asheville, N.C. Operating a mill at Erwin, Tenn., and grinding North Carolina feldspar.

The Cleveland Feldspar and Products Company (late Monteagle Feldspar Corp.), 327 Union Bldg., Cleveland, Ohio. Operating their own mine in Canada and grind Canadian spar in Cleveland.

Clinchfield Products Corporation, Erwin, Tenn. North Carolina spar.

Dominion Feldspar Corporation, Rochester, N.Y. Operating a mill at Genesee Dock, N.Y., and grinding Canadian feldspar exclusively. They also operate their own mine near Verona, Ont., under name of Feldspar Quarries, Ltd.

Erwin Feldspar Corporation, Erwin, Tenn. Operating a mill at Erwin, Tenn., and grinding North Carolina feldspar.

Eureka Flint and Spar Company, Trenton, N.J. Operating a mill at Trenton, N.J., and grinding Connecticut, New York, Canadian and North Carolina feldspar, both pure and blended.

Genesee Feldspar Co., Inc. (late Pennsylvania Feldspar Co.), Rochester, N.Y. Operating a mill at Genesee Docks, N.Y., which grinds Canadian spar, and a mill at Toughkenamon, Pa., which grinds Pennsylvania feldspar exclusively.

Golding Sons Company, Trenton, N.J. Operating mills at Trenton, N.J., Wilmington, Del., and East Liverpool, Ohio, and grinding Maine, Maryland, North Carolina and Canadian feldspar, both pure and blended.

Golding-Keene Company, Keene, N.H. Grinding New Hampshire spar.

TABLE II—*Concluded*

Feldspar Grinding Plants in Canada and in United States—Concluded

United States—Concluded

Name of Company—Concluded

Howe, L. W., South Glastonbury, Conn. Operating mill at South Glastonbury, Conn., and grinding Connecticut feldspar exclusively.

Maine Feldspar Company, Brunswick, Me. Operating mills at Topsham and Auburn, Me., and grinding Maine spars exclusively.

The Maryland Flint and Feldspar Company, Bel Air, Md.

Newell Mining and Pulverizing Company, Newell, W. Va.

Product Sales Company, 806 Equitable Bldg., Baltimore, Md. Operating a mill at Baltimore and grinding Maryland feldspar.

Potters Mining and Milling Company, East Liverpool, Ohio. Operating a mill at Loughlin, Ohio, and grinding Canadian and North Carolina feldspar.

Pennsylvania Pulverizing Company (New York Feldspar Corp.), Rochester, N.Y. One of the largest grinders of Canadian feldspar.

Rock Products Company, Nicholas Bldg., Toledo, Ohio. Operating a mill at Toledo and grinding feldspar from their own mine in Canada.

Southern Spar and Mica Company, 322 Haywood Bldg., Asheville, N.C.

Tidewater Feldspar Company, Middletown, Conn.

Trenton Flint and Spar Company, Brunswick, Me. Operating a mill at Cathance, Me., and grinding Maine feldspar exclusively.

Trenton Fireclay and Porcelain Company, Trenton, N.J.

United States Feldspar Company, East Liverpool, Ohio.

Wheeling Pulverizing Company, Wheeling, Va. Operating mill at Wheeling and used to grind Canadian spar.

Williams, Jr., Carl, Erwin, Tenn.

TABLE III

Dental Spar Shipments Since 1910

Year	Tons shipped	Value	Average per ton
		$	
1910	90	1,800	20 00
1911	12	240	20 00
1912	100	2,000	20 00
1913	74	1,554	21 00
1914	98	2,156	22 50
1915	32	640	20 00
1916	145	2,900	20 00
1917	220	5,300	24 09
1918	136	4,004	29 40
1919	352	10,208	29 00
1920	176	6,670	37 90
1921	64	2,080	32 50
1922	180	3,946	21 90
Total	1,679	43,498	26 00

Up to 1917 the output was obtained from the Villeneuve mine, Labelle county, Villeneuve tp., I, 30, Quebec.

1917. to date, the output was obtained from Villeneuve Extension, Labelle county, Portland E. tp. IX, 3, Quebec, and Glen Almond spar from the Derry quarry.

TABLE IV

Canadian Feldspar Shippers (over 50 tons), 1922

QUEBEC

Shipper	Address	Name of Mine	Location of Property			Local Office	Representative, Superintendent or Manager, etc.
			County	Township	Concession or Range Lot		
O'Brien and Fowler	Ottawa, 114 Wellington	Derry	Labelle	Derry	I, 7	Glen Almond	N. B. Davis, Mgr.
Mahoney and Rich Buckingham Feldspar Co.	Ottawa, 88 Bank St.		"	"	II, 9	"	E. Brewer, Mine Supt.
G. Pednaud	Buckingham		"	Buckingham	XII, S½ 14	"	G. Pednaud
Bush-Winning	N.D. de la Salette	Villeneuve Extension	"	Portland E.	IX, E½ 3	N.D. de la Salette	Bush-Winning

ONTARIO

S. H. Orser	Perth	Kirkham	Lanark	Bathurst	VII, 3	Perth	S. H. Orser
Rock Products	Toledo, Ohio, U.S.A.	W. J. Keays	"	"	IX, 20, 21	"	V. M. Gettins, Supt. A. G. Minehart, Mgr.
A. M. Campbell	Perth, Box 30	H. Keays	"	"	IX, W½ 20	"	M. P. Powers, Supt. A. M. Campbell, Mgr.
S. H. Orser	"		"	"	IX, 20, N½ 21	"	S. H. Orser
Monteagle Feldspar Co. Cleveland Feldspar and Products Co. Verona Mining Co.	Cleveland, Ohio, U.S.A.	Watson	Hastings	Monteagle	VI, 22, 23	Hybla	C. C. Burket, Secy.
Genesee Feldspar Co.	Rochester, N.Y., U.S.A.	McDonald	"	"	VII, 18	"	G. E. Worth, Mgr.
Feldspar Mines Corp.	Lewistown, Pa., U.S.A.	Woodcox	"	"	VIII, 16, 17	"	R. H. Thompson, Mine Supt. E. O. Banker, Gen. Supt.
Federal Feldspar, Ltd.	Ottawa, 46 Elgin St.		Frontenac	Bedford	III, 25	Tichborne	J. O'Toole, Supt. S. Fisher, Secy.
T. Craig	Verona		"	Portland	X, 3	Verona	T. Craig
Gardner and Treadwell	Hartington	Holleford	"	Loughborough	XI, 1	"	W. G. Gardner
Feldspar Quarries, Ltd.	Toronto, 60 Front St. E.	Imperial	"	"	XII, 1	"	W. H. Despard, Pres.
Can. Non-Metallic Minerals, Ltd.	Montreal, 207 St. James St.	Aylen Lake	Nipissing	Dickens	III, N½ 12, 13	Aylen Lake	W. Johnson
Industrial Minerals Corp.	Toronto, Bank of Hamilton Bldg.		Haliburton	Monmouth	XIV, 29	Monmouth	H. E. Harcourt
Industrial Minerals Corp.	Toronto, Bank of Hamilton Bldg.		Sudbury	Dryden	II, 9	Coniston	H. E. Harcourt
Wheeling Feldspar Co.	Wheeling, Va., U.S.A.	Cecebe Lake	Parry Sound	Chapman	II, 26	Burks Falls	W. B. Woods

V

FLUORSPAR IN 1922

V. L. Eardley-Wilmot

During the year Canadian fluorspar production amounted to 4,503 tons, valued at $102,138. In recent years practically all the fluorspar mined in Canada has come from the Rock Candy mine in British Columbia. In 1922 only 284 tons came from the Madoc district in Hastings county, Ontario, from which district in 1918 almost the entire Canadian output of 7,362 tons was won from some of the thirty, now dormant, mines.

Outside British Columbia the only two mines from which production was reported were the Wallbridge and McIllroy properties, and shipments from stock from the Perry mine, all in the Madoc district.

The Madoc fluorspar deposits mainly occur in veins occupying fault fissures in flat-lying beds of finely banded limestone and sandstone, and consist of pale green or yellow fluorspar, barite and calcite, intermixed in various proportions. The vein filling often occurs as a sand or gravel spar and a considerable proportion of the shipments during the year were in this form.

On the Wallbridge property in Madoc township, con. I, lot 1, the ore is very friable and is hoisted up a shaft which at the time of closing down was 55 feet deep. A platform has been erected at the shaft head upon which the ore is dumped, screened and hand picked. The larger lumps, after being washed by water in order to reveal the spar, are cobbed and further sorted. The fines go to waste.

The McIllroy mine in con. IV, lot 2, was operated for a few weeks by Mineral Products, Ltd. The old shaft was opened up, and material, mostly in the form of gravel spar, was shipped from the underground workings.

The Perry mine in Huntingdon township, con. VIII, lot 11, which is managed by Wm. Cross of Madoc, consists of extensive underground workings as well as several shafts along the strike of the vein for a distance of about 1,000 feet. The vein was first discovered in 1912 when the Grand Trunk railway was built, which exposed an outcrop, but little notice was taken of it at the time. No ore was mined during the year, but a few car loads were shipped from stocks.

The Madoc spar is used within the Dominion as a special flux for electric steel furnaces. The prices range from $13 to $15 per ton f.o.b. cars.

The Mines Branch at Ottawa has devoted considerable attention to the concentration of the Madoc fluorspar. Experiments in direct tabling, jigging, decrepitation, etc., were tried and rejected, and finally it was found that the best results were obtained by first calcining to slack the lime, and then tabling off the heavier barite. By this means a 95 per cent CaF_2 concentrate was produced.

The Rock Candy mine, owned by the Consolidated Mining and Smelting Company of Canada, is situated on Kennedy creek, a tributary of the north fork of the Kettle river, British Columbia. The nearest shipping point is Lynch Creek station, about 2 miles south of the concentrator.

The ore body, which outcrops on the side of a ravine, occurs on the almost vertical contact of an alkali syenite and a porphyry. The width of the deposit is very variable, being in some cases 140 feet wide, carrying veins of high grade fluorspar of a pale green colour. The vein matter consists mainly of quartz, while the occurrences of iron and copper sulphides are only in small segregations.

Mining has been carried out over a vertical distance of about 200 feet by means of three levels, which have been connected by raises and stopes. The most recent work consists of a series of drifts in the lower tunnel, the ore body being split by tongues of alkali syenite which have been utilized as pillars.

The mine is connected to the mill by an aerial tram of approximately two miles in length.

In concentrating the ore, advantage is taken of the fact that fluorspar when it is decrepitated splits up into minute fragments, while quartz is not so affected, so that after the heat treatment the products are merely screened to separate the fluorspar from its silica contents.

The Rock Candy mill flow sheet is as follows:—The ore is crushed in Blake crushers, screened over $\frac{1}{4}$ inch mesh, after which it passes through two sets of rolls and is then dried in a rotary dryer. The dried product passes over a series of impact screens from 8 to 15 mesh. The three sizes so produced are decrepitated in three separate 14 ft. x 3 ft. rotary kilns which make three revolutions per minute, at 1200° F. The 8-mesh products gives the best result, being under 2 per cent silica. The minus 15 is not decrepitated but regarded as a middlings product that passes over three vibrating tables in order to eliminate as much of the silica as possible. These table concentrates are dumped into the hot decrepitated products from the first two kilns in order to evaporate the moisture before screening. The decrepitated products are then separately screened. The combined smalls from the three kilns are sent to the shipping bins, while the oversize is rejected. These concentrates average 85 per cent CaF_2 and 6 per cent silica.

Although the mill was only operated for about four months during the year, 6,313 tons of ore were treated, and some of the rejects were put through the flotation plant at Trail. A total of 4,219 tons of high grade concentrates was produced, valued at $22 to $25 per ton. The material treated at Trail was converted into hydro-fluo-silicic acid used by the company in the process for the recovery of zinc.

H. E. Harcourt of Toronto has recently opened up a promising looking vein of deep purple fluorite in Haliburton county, Cardiff tp., con. XXI, lot 18, about 3 miles east of Wilberforce. No shipments have yet been made.

In the United States the main sources of supply come from Illinois and Kentucky. The new tariff of Sept. 1922 called for a duty of $5.60 per ton into the States.

The main use of the mineral is in the steel industry, mainly as a neutralizing agent and to increase the fluidity of basic open-hearth slags. The main demand is for a washed gravel of 85 per cent CaF_2 not exceeding 5 per cent silica free from sulphides of lead, zinc and barium. High silica means adding more limestone to the charge thereby increasing the duration of the time of heat and consequent reduction of tonnage.

The average consumption of fluorspar is roughly 8 to 10 pounds per ton of steel.

Next to the steel industry, the greatest use for fluorspar is in the ceramic, glass and enameling trades, where the highest grade hand picked and ground spar is used. The mineral is somewhat extensively used in electrolytic lead, antimony, etc., smelters. Its use in foundries is once more increasing. As sodium fluoride it is used for preserving wood.

In the United States the proportionate consumption of fluorspar is approximately as follows:—

Steel ingots and castings 80 to 85 per cent; Glass and enameling 7 to 10 per cent; Hydrofluoric acid 5 to 6 per cent; Foundries 1 to 2 per cent; Miscellaneous 2 per cent.

The steel industry in the United States has been steadily improving throughout the past year and this has been reciprocated in Canada, so that the future prospect for the Canadian fluorspar industry should be considerably brighter than in the past.

Canadian Fluorspar Statistics During Last Five Years

—	1918		1919		1920		1921		1922	
	Tons	Value	Tons	Value	Tons	Value	Tons	Value	Tons	Value
		$		$		$		$		$
Production—										
Ontario	7,187	150,779	3,425	59,281	3,758	68,475	116	1,744	284	3,905
British Columbia	175	5,250	1,638	38,556	7,477	171,971	5,403	134,523	4,219	98,233
Total	7,362	156,029	5,063	97,837	11,233	240,446	5,519	136,267	4,503	102,138
Exports			(a) 697	9,616	6,900	109,683	4,625	51,470	2,944	32,914
Imports—										
Hydro-fluosilicic acid	0·5	80	2·8	747	1·2	409	1·05	212	0·06	15
Fluorspar			(a) 8,273	84,702	6,812	113,818	3,867	43,752	4,980	73,343

(a) Last nine months.

65867—3½

VI

GRAPHITE IN CANADA, 1922

V. L. Eardley-Wilmot

The production of Canadian graphite experienced a marked decrease in 1918, due to the general reaction of business and an over-production during war years. The 1922 production of 650 tons is the lowest since 1908. Practically the whole of this output was produced by the Black Donald Graphite Co., Ltd., at Calabogie, Renfrew county, Ontario. During the year no ore was mined by that company, due to accumulated stocks of ore mined in 1919. Shipments of refined products since that date were the result of the treatment of these previously accumulated stocks of ore. Mining operations were conducted on a small scale by the Standard Graphite Company, at Guenette, Que., who mined and treated approximately 100 tons of ore, but made no shipments.

The Quebec Graphite Company, at Buckingham, Que., made some shipments of flake and manufactured products from stocks.

The Canadian market for flake graphite during the year was abnormally quiet. The ruling prices were extremely low, particularly for No. 1 and the lower grades, but the worst of the depression appears to have been reached about August, since which time there has been a gradual hardening in demand and a corresponding improvement in prices, which still continues.

The markets are to a certain extent governed by the productions from Ceylon and Madagascar. These countries have for the past few years been clearing up their enormous stocks produced during the war.

There was an increase in the Ceylon exports, but the stocks on January 1, 1923, amounted to about 15,000 tons, which shows a considerable reduction over the previous years, and prices have already stiffened up.

A continued demand for this material will necessitate the resumption of mining operations on the island, which have practically been at a standstill since 1918, due to over-production and lack of demand, which has caused extremely low selling prices that meant a heavy loss to the miners and producers. Ceylon importers are of the opinion that there will be a lapse of time before the mines are again operated on a large scale, and it is most likely that the graphite so produced will be marketed at a higher price than the after-war stocks that have mostly been sold at any price obtainable and much below the actual cost of production.

There was practically no production from Madagascar, but for the last three years about 6,000 tons annually have been exported from stocks. Madagascar flake is extensively used by the crucible makers in Great Britain, but in the United States it has not met with the same measure of popularity; hence comparatively little business was done in Madagascar graphite by the latter country, as crucible producers prefer Ceylon graphite.

There was a marked increase in the exports from Chosen, over 90 per cent of which was amorphous. Practically the whole of this latter material was shipped to Japan and was mostly re-exported from that country in the form of manufactured articles, particularly in the lead pencil industry.

About 35 per cent of the world's 1922 production came from Germany, this comparatively large output being mainly due to the depreciation of the mark which protected that country from strong foreign competition.

In the United States there was a slight increase in production, mainly due to the new tariff. There are practically no stocks of the domestic flake on hand at the mines, and during the year crystalline flake graphite was mined by three producers in Alabama, one in California, and one in Texas, while amorphous was mined by one company in Rhode Island and one in Nevada.

The following table gives the world's production of natural graphite for the last four years:[1]

World's Production of Natural Graphite, 1919-1922

(In metric tons)

	1919	1920	1921	1922
United States (*a*)—				
Amorphous	3,065	4,258	1,671	(*f*) 1,996
Crystalline	3,668	4,369	540	(*f*) 839
Canada (*a*) (*g*)	1,199	1,987	361	(*f*) 590
Mexico (*c*)	4,995	3,319	3,088	1,993
Austria	8,265	11,543	13,280	(*b*) 11,650
Czechoslovakia	31,234	(*b*) 15,000	(*b*) 12,000	(*b*) 13,000
Germany	30,525	20,639	(*b*) 30,000	(*b*) 35,000
Italy	7,626	5,398	(*f*) 5,750	(*f*) 7,300
Spain	1,958	6,315	2,772	(*d*)
Ceylon (*c*)	6,778	9,352	4,419	10,190
Japan	1,954	1,135	(*e*) 842	(*b*) 1,000
Chosen (*c*)	12,945	18,259	8,015	13,352
Madagascar	4,983	4,000	Negligible	Negligible
Other countries	8,442	1,574	(*d*)	(*d*)
	127,637	107,058 (*b*)	85,000 (*d*)	102,000

(*a*) Shipments. (*b*) Estimated. (*c*) Exports. (*d*) No data. (*e*) Unofficial. (*f*) Preliminary. (*g*) See Dominion Bureau of Statistics, p. 39.

The average price quotations during the last half of 1922 were steady as follows, f.o.b., New York:—

Ceylon, per pound			Madagascar, per ton				Mexican and Korean per ton		American flake per pound
Lump, 1st grade	Chip	Dust	Amorphous			Flake 85%	Crude amorphous	Higher grades	F.O.B. mines
			75-78%	80%	81-84%				
cts.	cts.	cts.	$	$	$	$	$	$	cts.
5-6	4-4½	3¼-3½	39	46	50	50	15-35	45-50	4-6

At the beginning of the new year the quotations for Ceylon lump and chip each rose one cent.

To the above quotations should be added the recently passed tariff on graphite going into the United States, which is 20 per cent on crystalline (Ceylon and Canadian), 10 per cent on amorphous (Mexican, Korean and Canadian) and 1½ cents per pound on flake (Madagascar, Canadian, etc.).

CANADIAN GRAPHITE OPERATORS

The deposit owned by the Black Donald Graphite Co., Ltd., occurs on Whitefish Lake near Calabogie, con. III, lots 17 and 18, Brougham tp., Renfrew county, Ontario, and constitutes the richest graphite ore body on the North American continent.

[1] "Eng. and Mg. Jour-Press," p. 976, June 2, 1923, by A. H. Redfield, U.S. Geol. Surv.

The country rock consists of a coarse limestone in which are interlaced bands of hornblende schist. The ore body itself has been subjected to extreme deformation and folding which in places assumes the nature of a blanket vein. This graphite ore is of a dense micro-crystalline structure, some parts of which are classed as amorphous, while other portions of the ore body are made up of scattered particles of crystalline.

The vein has been followed by inclined shafts and extensive workings under the lake for several hundred feet at a depth of 200 feet.

During the year no mining was done, but the supply of crude ore was drawn from large reserve stocks mined and accumulated soon after the cessation of the war. R. F. Bunting, the president, states that he has planned to commence mining operations in the spring of 1923 on the blanket vein which outcrops behind the mill and thus avoid having to dewater the main workings under the lake.

Seventeen hundred tons of crude ore from yard stocks were milled in the up-to-date concentrator at the plant, which consists of stamps, Callow oil flotation units, screens, settling tanks and dryers. The finishing department consists of buhr stones and screens whereby the material is finally classified into seven different products. The highest grade, No. 1, analyses 97 to 99 per cent graphitic carbon, and the lowest grade dust about 65 per cent. The mill feed varies between 55 and 65 per cent. The main output is in the form of dusts or plumbago, which are extensively used for foundry facings. Although the proportion of No. 1 is small, it is, on account of the quantity produced, quite considerable. It takes about 400 H.P. to operate the whole plant, which is obtained from the company's power house on the Madawaska river.

The property of the Standard Graphite Company, of Montreal, is located on ranges VI and VII, lots 27-30, Boyer tp., Labelle county, Quebec, two miles east of Guenette station, which is 143 miles north of Montreal. A spur line runs through the property.

The rocks consist of biotite gneiss, which shows the nearby influence of limestone, and are occasionally intersected by soda bearing pegmatite dikes. The whole region is somewhat distorted and metamorphosed. The ore zone itself has a considerable areal distribution, in which the graphite occurs as more or less continuous series of lenses of varying width which follow the general strike and contortions of the rocks in which they occur. Both the rocks and the type of graphite are typical of the majority of graphite deposits in the Buckingham district.

The main ore zone, to which the present workings are confined, shows a surface width of 8 feet of good ore, and outcrops along its strike for at least 200 feet. Besides this there are in the immediate vicinity a number of approximately parallel and narrow ore shoots of one to three feet in width. The main ore body has been stripped and quarried over about 100 feet.

The graphite, which carries an appreciable amount of mica, has in places penetrated several feet into the footwall, but is of low grade. The higher grade milling ore runs about 20 per cent graphitic carbon, which, with the exception of the Black Donald deposit, is considerably above the average of the Canadian graphite ores.

A concentrator has been erected on the property close to the railway spur, about a quarter of a mile from the workings, and consists of a Spearman film flotation unit which was run for a period of two months in the summer, when 100 tons were treated. Changes of management have, however, recently taken place, and the mill flow sheet is being altered and Callow oil flotation units are being installed. It was originally intended to obtain motive power from a lake 500 yards distant, but construction work on this has been temporarily abandoned.

The property is now being worked by E. L. Goyette, of the Finance and Industries, Ltd., Montreal, by an arrangement with the Standard Graphite Company.

In the Buckingham district no ore was mined or milled by the Quebec Graphite Company during the year, but material in the form of flake, foundry facings and lubricating graphite in tins was sold from stocks.

The old North American Graphite Co., Ltd., whose property is located eight miles west of Buckingham, has been reorganized. H. P. H. Brumell states that he intends to remodel the old concentrator (which was erected in 1895) and install a Callow oil flotation system using flat shallow cells. The mill feed runs between 10 and 12 per cent graphite.

Early in the year the Consolidated Graphite Company's property, seven miles north of Buckingham, was bought by J. H. Cameron of that town. The property includes the mine, concentrator with machinery, and several hundred tons of mixed concentrates.

The outlook for the domestic industry appears much brighter than in the past, more particularly for high grade dusts, or plumbago, rather than for flake and lower grades. It should nevertheless be borne in mind that an increase in the demand and price of graphite is liable to lead to increased production of both foreign and domestic material, and thus cause further competition.

Canadian Graphite Statistics for Last Three Years[1]

	1920		1921		1922	
	Tons	Value	Tons	Value	Tons	Value
		$		$		$
Refined graphite shipments—						
Flake	421	68,954	149	29,187	95	16,170
Dusts	1,769	96,663	788	36,675	555	19,454
Total	2,190	165,617	937	65,862	650	35,624
Exports—						
Concentrates—Plumbago and flake	2,142	159,817	614	40,809	452	16,619
Imports—						
Plumbago—crucibles	.....	(a)176,717	.....	23,786	.....	36,961
Plumbago—not ground or manufactured	.....	4,352	.....	4,141	.....	1,007
Plumbago—ground and manufactured	.....	102,568	.....	47,463	.....	47,095
Total imports	.....	283,637	.....	75,390	.....	85,063

(a) Includes clay crucibles.
[1] Dominion Bureau of Statistics.

VII

TALC AND SOAPSTONE IN CANADA, 1922

V. L. Eardley-Wilmot

During the year there was a slight increase of talc production over 1921, but the output is still considerably below the normal. There were seven companies that reported activity, four of which were shippers. Seventy-five per cent of the 14,500 tons of talc mined in 1922 was mined by the Henderson Mines, Ltd., at Madoc, in Huntingdon tp., XIV, lot 14.

This mine has been in operation since 1899 and constitutes one of the world's important producers of superfine white foliated talc. The deposit occurs in a quartzose crystalline limestone of Grenville age, and has a width of 25 to 40 feet and is in the form of a flat horseshoe, due to the strata having been folded. The extent of the deposit is undetermined. The workings consist of a large open-cut and about 2,000 feet of driftings and cross-cuts, mainly on the 150 and 200 foot levels. During the year the new shaft was completed and connected by a cross-cut to the 200 foot level, also considerable drifting was done in an easterly direction from the end of the same level.

All the ore is hauled one mile by teams to the mill at Madoc station owned by Geo. H. Gillespie. The material goes through a series of grindings and screenings whereby three grades are produced, which are valued at $9 to $22 per ton according to grade.

The greater part of the remainder of the Canadian talc output was produced by the Asbestos Pulp Company, whose property, known as the Connolly mine, is on the adjacent lot east (lot 15) of the Henderson mine, and is a continuation of the same ore body.

The workings consist of about 1,500 feet of drifts and cross-cuts on four levels. During the year a new shaft 110 feet in depth was sunk at the east end of the workings and connected with an intermediate level, from which some cross-cutting to the north has been conducted in the nature of prospecting work. Considerable cross-cutting and drifting has also been carried out in a southerly direction from the west of the main shaft at the 185-foot level.

A mill was built on the property about six years ago by the Anglo-American Talc Corporation, and consists of the usual grinding and screening units, with the addition of an air float chamber lined with canvas.

The highest grade of refined talc from the Madoc district is used as talcum powder, while the other grades are used as fillers for the paper and rubber industries.

About a car lot of talc was mined by H. H. Wood at Mine Centre in the Rainy River district, Ontario. Crayons for metal workers and heat resisting and electrical insulators were manufactured by means of a sawing, lathing and baking process.

The Wabigoon Soapstone Company has opened up a deposit in Zealand township in the district of Kenora, Ontario.

The Megantic Mining Company did a little prospecting work at New Ireland, Megantic county, Quebec, and preparations are being made to install machinery during 1923.

It is of interest to note that some of the material from the soapstone deposits of the asbestos district is now being quarried and cut into bricks, which are being used as a lining for alkali recovery furnaces in paper-pulp mills. Although the production of these bricks so far has been very small, there is, nevertheless, an important consumption of the imported product, which may in future be supplied from within the Dominion.

In British Columbia two companies mined and shipped a few hundred tons of talc. The Canadian Talc and Silver Company did some development work during the late fall and commenced erecting a mill at Keefers P.O., in township XII, range 26.

A few car lots of ground talc were shipped to Vancouver and Victoria by the Eagle Talc and Mining Company, Ltd., from their deposit at Wolf creek, Victoria mining division, B.C.

Canadian Talc Statistics for the Last Five Years

	1918		1919		1920		1921[1]		1922[2]	
	Tons	Value	Tons	Value	Tons	Value	Tons	Value	Tons	Value
		$		$		$		$		$
Production—										
Crude	12,772	47,494	12,243	49,074	11,820	48,939				
Refined	5,397	71,703	6,399	67,221	9,851	117,995				
Total	18,169	119,197	18,642	116,295	21,671	166,934	10,134	144,565	13,558	185,197
Exports—										
Crude	.....	208,301	.....	210,150	.....	10,653	71	437	Nil	Nil
Refined	.....		.....		14,909	253,055	7,008	112,053	9,854	143,938
Total refined sold[2]	15,903	222,167	15,927	235,000	19,610	303,738	10,063	144,128	13,058	182,697

[1] Practically all refined talc. [2] Product Canadian plants.

VIII

THE MOLYBDENUM SITUATION IN CANADA, 1922

V. L. Eardley-Wilmot

War requirements greatly stimulated the demand for molybdenite, and in Canada the maximum output was reached in 1918, when 34,000 tons of the ore were mined from which concentrates containing 189 tons of pure molybdenite were produced. The price of concentrates was controlled at $1.09 per pound at Ottawa.

Although there has been no actual molybdenite production in Canada for the last three years, in 1922 the following companies reported a certain amount of activity in the way of further prospecting and constructional work, and are preparing for production in the near future:—

The Molybdenite Reduction Company, known as the Benjamin mine, in the Abitibi district, Quebec, L. N. Benjamin, Managing Director, 108 Laurier Ave. W., Montreal, Que.

St. Maurice Mines Company, on the Indian peninsula, Kewagama lake, in the same district as above, P. Geddes Grant, Director, 342 Madison Ave., New York city.

The United Molybdenum Corporation, known as the Joiner mine, at Wilberforce, Haliburton county, Ontario, W. E. Joiner, Director, 76 Sun Life Bldg., Toronto.

American Molybdenites, Ltd., three miles west of Joiner mine, W. J. L. McKay, Managing Director, 42 Yonge St. Arcade, Toronto.

The Indian Lake property, known as the Bain mine, west of the Gatineau, in Masham township, Quebec, H. H. Claudet, Manager, 363 Sparks St., Ottawa, Ont.

It is felt that one cause of the inactivity in the molybdenum industry is that the steel maker or consumer of the metal hesitates to make contracts until he is assured of his supply of the raw material, and the producer will not commence mining operations until he is assured of the disposal of his output and at a price satisfactory to him.

As compared to most other minerals, molybdenite deposits are scattered and of low grade, generally between one-half and one per cent, so that production costs per pound of high grade concentrates (85 to 90 per cent MoS_2) are high. The nominal market quotations of the mineral, particularly in Europe, are so low that the miner is not assured of profits and hesitates to commence producing. There are, however, a number of deposits in Canada from which a very considerable quantity of the mineral can be produced as soon as the demand and market prices improve.

In the recent United States tariff revision the following amendment was passed by the 67th Congress, in August 1922. The clauses of this tariff affecting molybdenum and molybdenum products are as follows:—

> Molybdenum ore or concentrates, 35 cents per pound on the metallic molybdenum contained therein.
>
> Ferro-molybdenum, metallic molybdenum, molybdenum powder, calcium molybdate and all other compounds and alloys of molybdenum, 50 cents per pound on the molybdenum contained therein and 15 per centum ad valorem.

The duty on an 85 per cent MoS_2 concentrate is, therefore, approximately 18 cents per pound.

The most recent New York quotations are 65 to 75 cents per pound of MoS_2 in an 85 per cent MoS_2 concentrate, including duty.

In the United States this alloy is produced by at least a dozen different firms.

Small quantities of molybdic acid have been produced by J. W. Evans, manager of the Tivani Electric Steel Company, in his laboratories at Brockville, Ont. This company during the war produced considerable quantities of ferro-molybdenum, and it is understood that they are ready to continue the manufacture of this alloy as soon as the demand arises.

A number of automobile firms in the United States are using molybdenum in parts of their cars, and recently a molybdenum steel shovel and molybdenum rolls have been put on the market for which highly satisfactory results are claimed.

The present day uses are mainly for what are known as low molybdenum alloy steels for structural and engineering purposes, in which only one-quarter of one per cent of molybdenum is used in combination with other alloying metals.

Several Canadian producers of the mineral realize the future possibilities and are making preparations for production, but for the reasons already stated they require further stimulation from the consumers.

IX

BITUMINOUS SANDS OF NORTHERN ALBERTA

S. C. Ells

An extensive deposit of bituminous sand—commonly though incorrectly referred to as "tar sand"—outcrops at frequent intervals along the Athabaska river and its tributaries through an aggregate distance of approximately 200 miles, in the district centering about McMurray. Unfortunately a great deal of misconception regarding the character and extent of this deposit has arisen through exaggerated and otherwise incorrect statements by uninformed people.

The bituminous sand of Alberta is at present attracting the attention of responsible persons. This may be attributed, in part, to the recent completion of the Alberta and Great Waterways railway to the provisional head of navigation at Waterways, and, in part, to legitimate publicity given the deposit through the Mines Branch.

The area represented by outcrops, and presumably underlain by bituminous sand, may be arbitrarily defined as lying between W. long. 110° and 113° and between N. lat. 56° and 58°. All exposures within this area lie within a radius of 60 miles of McMurray. Upwards of 250 exposures, all of which represent portions of one continuous deposit, have been examined and measured. The direct distance in a north and south direction through which outcrops have been noted is approximately 110 miles, and that from east to west approximately 80 miles.

Certain of the above outcrops represent portions of a deposit that with reasonably favourable market and transportation conditions, will eventually prove commercially valuable. But it is also true that a large portion of the area underlain by bituminous sand must be considered as of very doubtful economic value. This statement is based on a consideration of controlling factors such as thickness, character, and possibility of disposing of overburden, transportation, percentage of associated bitumen and uniformity of material. Each of these factors should be given careful and detailed study in considering commercial development in any portion of the McMurray district. Other important, though subsidiary factors, are fuel, labour, water supply, and climatic conditions.

Associated with the bituminous sand are occasional small seepages of bitumen, which originate in the richer beds, and are locally known as tar springs. None of these is of sufficient extent to be considered of economic importance, nor can they be considered as indicating the probable presence of a petroleum pool within the area under consideration.

The outstanding features presented by the McMurray deposit may be briefly stated as follows:—

(*a*) That the deposit represents the largest known occurrence of solid asphaltic material.

(*b*) That the deposit is, as yet, totally undeveloped.

(*c*) That at the present time practically all asphaltic materials used in Canada are imported from foreign countries.

Three possible lines along which the bituminous sand may be commercially developed are suggested:—

1. That the crude material may be used in the surfacing of streets and highways. In 1915, in order to demonstrate the merits of the material, the writer designed and laid in Edmonton, Alberta, areas of standard sheet asphalt, bitulithic and bituminous concrete. This effort marked the first attempt to commercially utilize the bituminous sand as a paving material, and the result was entirely satisfactory. The extent to which the crude material may be used for such purposes will be largely determined by freight charges. In competition with imported asphalts and under existing freight rates the permissible freight haul for crude bituminous sand will probably be limited to about 625 miles.

2. That the bitumen, which is of a high grade, be separated from the sand aggregate, and utilized for a number of recognized purposes for which such material is well adapted.

The results of the writer's research work, in 1916, in connection with the separation problem, although encouraging, are as yet not conclusive. This work included separation by means of various solvents, by the use of centrifuges, by the use of water heated under pressures ranging up to 92 lbs. per sq. inch, followed by filtration, and by the use of flotation cells. Results from the use of flotation, using heated water to which certain reagents had been added, appeared to give the most favourable results.

3. That the crude bituminous sand be retorted with a recovery of crude petroleum. Such a distillation was made by the writer in 1915, using a retort with a capacity of 25 pounds crude bituminous sand. The crude petroleum derived in this manner was then fractionally distilled and

the various distillates refined. Possibilities of such distillation on a commercial scale have yet to be determined, but there are strong indications that the associated hydrocarbons can be successfully recovered from the bituminous sand in this manner.

It has been recognized for many years that large commercial development of the bituminous sand will depend on the successful recovery—either by separation or distillation—of the associated hydrocarbons. Such a recovery presents a problem which is apparently not more difficult than many metallurgical problems which have already been successfully solved. The problem will be solved only by those methods which have been successfully applied in metallurgical work—intelligent research, mechanically sound design, and careful, correlated experimentation.

Successful commercial development will depend on making no false moves during initial stages of development, and in having no "lost motion" in subsequent operations. Consequently, as a preliminary step every effort should be made to secure the most complete information possible regarding the deposits.

CHEMICAL DIVISION

I

SOME CANADIAN FOSSIL RESINS

R. T. Elworthy

Fossil resins have been known to exist in certain districts in western Canada since the earliest explorations of prospectors and geologists. Results of the examination of samples from the north Saskatchewan river, from Peace river and from the Nechacco river south of Fort Fraser, B.C., were recorded in the report of the Geological Survey for 1876-77, by Dr. Harrington. Other localities where such fossil resins have been found, usually associated with lignites, are the Queen Charlotte Islands, B.C., in Saskatchewan, the Francis river in the Yukon, and in many parts of British Columbia.

Recently a considerable quantity of fossil resin, which was thought to be amber, was recognized in the waste dumps at Coalmont collieries, Coalmont, B.C. In view of the possibilities if it proved to be amber, an investigation of the material and of methods of separation from the coal was carried out in the Mines Department.

This report describes the results of the work undertaken in the Chemical Division, on the physical and chemical properties of this resin, in which the writer was ably assisted by R. J. Offord, Sr. Lab. Asst. Methods of separation from the coal are described in the Ore Dressing Division report.

A sample of fossil resin from Cedar lake, Manitoba, was also examined. This came from the same locality as that reported on by Dr. Harrington in 1901, and proved to vary little from the material studied at that time.[1]

Tests were also made to determine the possibility of their industrial utilization.

For purposes of comparison a short account of amber and its properties precedes the description of the laboratory investigation.

AMBER AND ITS PROPERTIES

Amber is a hard fossil resin, varying in colour from lemon yellow to deep brown, sometimes transparent but usually translucent or opaque. It was prized among the ancients as a material for making personal ornaments and was of interest to the early philosophers on account of its property of becoming electrified by rubbing, which gave it the power of electrical attraction. The oil obtained by the destructive distillation of amber was valued as a perfume. The chief source of the world's supply of amber has been the shores of the Baltic, especially from mines along the coast of East Prussia.

The mineral occurs in nodules disseminated in clay of the Cretaceous formation or associated with lignite, and when liberated by the action of the waves, is washed up on the sea shores, where it is collected by hand. Later developments led to the digging of mines, some even under the sea, and to the installation of dredging and washing machinery. The

[1] Harrington, Am. Jour. Sc., 1891, Vol. 42.

production of the material is now a government monopoly and thousands of pounds were produced annually before the war. The chief use was in the manufacture of beads and ornaments and for pipe stems and cigarette holders. Years ago it was highly prized as the basis of varnishes, but it has been displaced by the cheaper eastern copals.

Other varieties of fossil resins have been found in Germany, Austria, Russia, Poland, India, Burma, Japan and Canada. Mineralogists have endeavoured to classify these separate occurrences. Succinite, retinite, gedanite, glessite, stantienite, backerite, chemawinite, etc., are a few of the species described It is the usual practice to apply the name amber to the particular variety succinite, which is the chief material obtained in Prussia. It differs from the other varieties in its higher yield of succinic acid when it is destructively distilled. In view of the complex nature of these fossil substances, which have been formed from balsams and resins subjected to age-long processes of condensation and volatilization, constancy of composition and identity of properties such as is found in the mineral kingdom cannot be expected.

Although the physical properties and analytical constants have been determined for many of the fossil resins, comparatively little work has been done in the investigation of the chemical nature of the compounds composing them. The outstanding investigator is Prof. A. Tschirch[1], who has shown that the chief components of Prussian amber are free succinoabietic acid, esters of this acid and succino-resene, a substance of unknown constitution. The difficulties of separating, identifying and determining the constituents of these resins are considerable.

THE COALMONT FOSSIL RESIN

Occurrence.—The material subjected to examination in the laboratories of the Mines Branch came from the Coalmont collieries, Coalmont, B.C., where it occurs associated with the lower grade coal close to the slate contact of the coal seams. It was first observed in the waste dumps. The material is found in the coal in thin seams about a quarter of an inch thick, or as isolated fragments. Pieces larger in diameter than one-fourth of an inch are seldom present. For complete separation the coal has to be crushed considerably smaller than this mesh. The small size of the resin particles thus obtained is one of the handicaps in its commercial utilization.

The material used for the following tests was the product recovered by separation from four hundred pounds of the coal in the Ore Dressing Laboratories during the progress of concentration tests carried out by the division in their experiments.

THE CEDAR LAKE RESIN

This material was provided by Messrs. J. T. Donald Co., of Montreal, and was part of a sample obtained from Saskatchewan by Mr. J. Kemp of Montreal. It was collected from the shores of Cedar lake on the main Saskatchewan river, and like the earlier occurrences reported contained much debris and fragments of wood.[2]

[1] A. Tschirch. Harze und Harzebehalter. Leipsig. 1906.
[2] J. B. Tyrell, G.S.C., 1890-91, Part E.

EXAMINATION OF PHYSICAL PROPERTIES

Appearance.—The pieces of Coalmont resin, separated from the coal, are irregular in shape with no marked cleavage lines. The material is brittle and breaks with a conchoidal fracture. The colour varies from light lemon yellow through all shades of brown to dark green and even black. Most pieces are transparent to translucent; the darker pieces are opaque. A light green fluorescence is often noticeable.

The Cedar Lake material is brown and more uniform in colour. The particles vary in size from one-tenth to one-half inch in diameter, and hav ng been subjected to the action of the waves and weather are more rounded and regular in shape.

Hardness.—The hardness of pieces of Coalmont resin varies between two and three. Most fragments will scratch gypsum (hardness 2) but not calcite (hardness 3). The Cedar Lake specimens gave similar results.

The hardness of amber is given as 2 to 2·5.

Specific Gravity.—Determinations were made by weighing and by ascertaining the densities of solutions in which pieces just floated. The results for different pieces of Coalmont resin varied considerably. The limits were 1·031 to 1·168. The Cedar Lake material gave similar results. This variation is due chiefly to occluded gas bubbles and is common to substances of this nature. The specific gravity of Prussian amber is given by Dana (Descriptive Mineralogy, p. 1002) as 1·050 to 1·096.

Similar tests on pieces of the coal from which the Coalmont resin had been separated gave 1·308 to 1·313, showing much less difference between individual fragments.

TESTS ON THE WASTE COAL FROM COALMONT WHICH CONTAINED THE FOSSIL RESIN

It was necessary to find the amount of resin in the coal and to determine the size to which it would be necessary to crush it in order to free the greater part of the resin from adhering coal.

METHODS OF ANALYSIS

Two methods were tried to determine the amounts of resin and coal in the samples at different stages of treatment.

Counting Method

The first method used was to count the number of pieces of resin and of coal in a representative sample of uniformly sized particles, under the microscope.

A portion of the sample was spread on a slide on which had been engraved with a diamond a checker board of small squares. Using a combination of objective and eyepiece, which gave a magnification of 25, with transmitted light, the particles of coal and resin were sharply differentiated, and if not spread too thickly could be readily counted.

Two slides were used, one with 25 squares, each one-tenth inch size, best for particles between 28 and 100 mesh, and one with 15 squares, each of 5 mm. size for particles 10 to 28 mesh.

The following example shows the method of working:—

Counts	Resin	Coal	Per cent by volume		Per cent by weight	
			Resin	Coal	Resin	Coal
(1.) 20 squares	205	53	79·5	20·5	76·4	23·6
(2.) 20 "	143	37	79·5	20·5	76·4	23·6
(3.) 20 "	263	91	74·3	25·7	70·6	29·4
(4.) 20 "	200	56	78·2	21·8	74·9	25·1
Mean			77·9	22·1	74·5	25·5

The weights were calculated by assuming the particles counted to be of equal volume and by multiplying respectively by 1·08 sp. gr. of resin and 1·30 sp. gr. of coal. This mixture was artificially made up of 78·5 gm. amber and 21·5 gm. coal. The agreement is fair.

Other tests were made on mixtures containing less resin, both by counting and by actually separating the amber and coal mechanically and weighing the portions thus separated.

	−20 and +28 mesh		−28 and +35 mesh	
	Coal	Amber	Coal	Amber
By separation and weighing	98·1	1·9	98·5	1·5
By counting	97·7	2·3	97·4	2·6

The method is only suitable for particles between about 20 and 65 mesh.

Flotation Method

Experiments with solutions of salts such as sodium chloride and calcium chloride of specific gravity about 1·10 to 1·20 showed that clean separation of the coal and resin could be made by shaking up known weight of the mixtures with such solutions, allowing them to stand and then removing the floating resin or the sunken coal.

For samples smaller than 20 mesh a separatory funnel served well. Five to fifty grams of the mixture was placed in the funnel, and 50-100 c.c. calcium chloride solution, sp. gr. 1·20 added. The funnel was well shaken for several minutes to remove all air bubbles on the particles, and then left from one to five hours for the coal particles to settle, the time depending on the size of the particles. The coal was then run off through the stop cock, and usually more solution added to the residue and the mixture again shaken up and left. Two or three such treatments gave almost complete separations. The two portions were then filtered, well washed, dried and weighed and the proportions calculated.

For larger mesh samples, a pear shaped glass vessel closed at the end by a rubber tube and clip made it easy to run off the larger pieces. This method is more accurate, though not quite as rapid, as the counting method.

ESTIMATION OF AMOUNT OF RESIN IN HEAD SAMPLE

A representative 100 gm. sample of the coal was taken and the amount of resin determined by the methods just described. The weight of resin separated was 1·72 gm.

The screen analysis, described in the next section, gives the range of sizes of the pieces in this sample.

To determine the effect of crushing to a smaller size some of the sample was ground to pass 100 mesh, but not 150, and the resin content then determined. Separation by floating in calcium chloride solution was not readily attained, and a number of treatments were necessary to obtain a good product. From 25 gms. 0·493 gm. was recovered equal to 1·96 per cent. Crushing to this small size would not be practical in a commercial separation process.

SCREEN ANALYSIS OF HEAD SAMPLE

Two hundred grams of the head sample as received from the ore dressing plant and as used in flotation experiments was shaken in a series of Tyler standard screens in a mechanical shaker with the following results:

Screen scale ratio, 1·414.

Openings			Diameter wire inches	Weights		
Inches	Millimeters	Mesh		Sample weights	Per cent	Per cent cumulative weights
0·065	1·651	10	0·035	79·85	39·92	39·92
0·046	1·168	14	0·025	20·70	10·35	50·27
0·0328	0·833	20	0·0172	19·20	9·60	59·87
0·0232	0·589	28	0·0125	17·85	8·92	68·79
0·0164	0·417	35	0·0122	13·65	6·82	75·61
0·0116	0·295	48	0·0092	12·90	6·45	82·06
0·0082	0·208	65	0·0072	8·95	4·48	86·54
0·0058	0·147	100	0·0042	8·35	4·18	91·72
0·0041	0·104	150	0·0026	2·75	1·37	92·09
0·0029	0·074	200	0·0021	6·50	3·25	95·34
0·0029	0·074	200	0·0021	8·40	4·20	99·54

ESTIMATE OF RESIN IN EACH SCREEN SIZE OBTAINED IN THE SCREEN ANALYSIS

The eleven fractions of the 200 gm. head sample used on the screen analysis were then analysed for their content of resin. The fractions staying on 10 and 14 mesh were analysed by mechanical separation of the resin and weighing. The estimation of the resin content of the finer mesh samples was carried out by the flotation method.

The following results were obtained:—

Mesh	Opening millimeters	Sample weight	Per cent	Weight	Per cent
10	1·651	79·85	39·92	0·73	0·91
14	1·168	20·70	10·35	0·16	0·77
20	0·833	19·20	9·60	0·36	1·50
28	0·589	17·85	8·92	0·27	1·51
35	0·417	13·65	6·82	0·34	2·5
48	0·295	12·90	6·45	0·24	1·9
65	0·208	8·95	4·48	0·33	3·7
100	0·147	8·35	4·18	0·40	4·8
150	0·104	2·75	1·37	0·09	3·9
200	0·074	6·50	3·25	0·24	3·7
200	0·074	8·40	4·20	0·08	1·0
		199·10		3·24	

=1·63 per cent

This checks well with the results obtained by straight flotation 1·72 per cent.

As might be expected more resin is found in the smaller mesh samples as there is more chance of a greater number of pieces being present, free from attached coal when they will float.

THE GRADING OF THE RESIN BY MEANS OF SOLUTIONS OF VARIOUS DENSITIES

On account of the considerable range of specific gravity of the resin it seemed of interest to see what the result would be from treating a quantity of the pure product with solutions of different densities. For this purpose 20 gms. of resin free from coal and varying in size from 20 mesh to 65 mesh was treated with the following solutions and that which floated was separated and weighed. The table shows the percentage floating in each solution and the cumulative percentage:—

	Weight	Per cent	Total floating per cent
(*a*) Floats on distilled water sp. gr. 1·00	0·0396	0·19	0·19
(*b*) " solution " " 1·03	0·7868	3·93	4·12
(*c*) " " " " 1·05	7·1304	35·65	29·77
(*d*) " " " " 1·08	3·9357	19·68	59·45
(*e*) " " " " 1·10	2·1193	10·60	70·05
(*f*) " " " " 1·15	3·0232	15·10	85·15
(*g*) " " " " 1·20	0·8345	4·16	89·31
(*h*) Residue sinks " " 1·20	0·7842	3·92	
Lost in separations	1·3493	6·77	
	20·0000	100·00	

These figures well illustrate the variation in specific gravity of the resin. When the separate portions *b*, *c*, *d*, etc., were compared a very satisfactory gradation in colour was noted. Although *b* was not wholly free from dark particles the general colour was light brown. Portion *d* was perceptibly darker than *c* and lighter than *e*. There was little difference between *e*, *f* and *g*.

Larger quantities of the resin have been separated into three grades in this way:—

Grade 1. Floats on solution sp. gr. 1·03.
Grade 2. Floats on sp. gr. 1·09 and sinks in sp. gr. 1·03.
Grade 3. Sinks in sp. gr. 1·09.

The difference in colour and transparency is very marked. It is evident that this would serve as a commercial means of grading the product. With narrower limits of density even better classification would be possible.

SOLUTIONS FOR FLOTATION AND GRADING

The following table gives the specific gravities of various strength solutions of some common soluble salts:—

Grams substance in 100 gm. solution.

——	2	4	6	8	10	12	14	16	18	20	22	24	26	35	45
Sodium chloride.....	·011	·025	·039	·053	·038	·083	·099	·114	·130	·146	·164	·181	·197*		
Sodium sulphate.....	·016	·035	·053	·072	·091	·110	·130	·145*	....	....	....	....	.....		
Magnesium sulphate..	·018	·039	·060	·081	·103	·125	·148	·170	·194	·217	·242	·266	·292*		
Calcium chloride.....	·014	·031	·048	·065	·082	·100	·118	·137	·166	·175	·196	·214	·235	·329	·442*

*Saturation point at 20° C.

These figures are the decimal part of the value for the specific gravity. The deposits of alkali salts now under investigation in the west might prove a cheap source of sodium and magnesium sulphates for local use.

Although there is not a very great difference between the specific gravity of the same percentage solution of sodium chloride and calcium chloride, the percentage of saturation is very much less in the case of the calcium chloride on account of its greater solubility.

The practical advantage of this is that solutions of 10-15 per cent of calcium chloride when spilt or splashed will crystallize less readily than 10-15 per cent sodium chloride and would be cleaner to work with. On the other hand sodium chloride is cheaper ($10-15 per ton) than calcium chloride ($20-25 per ton).

CHEMICAL PROPERTIES

In the following paragraphs the results of the determinations of the various analytical constants are reported, especially with a view to comparing the fossil resins with amber. No separations of the constituents or attempts to determine the constitution have yet been made.

Softening and Melting Points

Prussian amber begins to soften at about 150° C. at atmospheric pressure. Various figures have been reported for the melting point.

Authority	Reference	Melting point ° C.
Thorpe.....	Dictionary of Applied Chemistry, page 183, Vol. I, 1921.....	350–375
M. B. Blackler.....	Allen's Commercial Organic Analysis, page 19, Vol. IV, 1915.....	287
Livache and McIntosh.....	The Manufacture of Varnish and Kindred Industries, page 134, Vol. II, 1920.....	250–300
Morrell and de Waele.....	Rubber, Resins, Paints and Varnishes, page 86, 1921.	280–315

The following table gives the results obtained with the Coalmont resin;—

Weight	Size	Softens °C.	Melting point °C.	Oil distilled per cent by weight
(*a*) 5 gm	10 mesh	180	250–270	76
(*b*) 25 "	8–35 "	200	270–297	60

A sample of Prussian amber, treated in the same way, gave:—

(*c*) 10 gm		160	260	

The results of a similar test on the Cedar Lake resin were:—

(*d*) 25 gm	3–10 mesh	120	220–240	75·6

The Products of Distillation

A number of specimens of different samples of amber, obtained from various sources, as well as the Cedar Lake and Coalmont resins, were subjected to destructive distillation.

In the case of amber the initial products are water and a light lemon yellow coloured oil. As the temperature rises the oily distillate becomes darker in colour. The vapours have a characteristic odour.

The Coalmont and the Cedar Lake resins gave very similar results though less water was collected. The oils closely resembled amber oil. The Coalmont resin oil had a specific gravity of 0·939 at 20°C. It was readily soluble in ether, chloroform and carbon tetrachloride, but only partially soluble in absolute alcohol. It decolorized a solution of bromine in carbon tetrachloride, showing it to be unsaturated.

It formed an oily nitro compound on nitrating in the usual way. Alkalies darkened and resinified it. On redistillation of the crude distillate three fractions, 140-180°C., 280-310°C., and 350-370°C., were obtained, the first light yellow, the second pale green, and the third brown in colour. All the distillates darkened slowly on standing in closed vessels and very much more rapidly when exposed to the air. A dark brown resinous residue forming 20 per cent of the original oil remained after distillation. It was soluble in turpentine and in gasoline.

When distillation was carried to completion, the residue was a black tarry substance only slightly soluble in turpentine and linseed oil. Distillation yielded about 70 per cent of the resin as oil. On the other hand heated to about 300°C., when it is well molten as it would be in the process of varnish manufacture, the resin lost about 12 per cent of its weight. This figure agrees[1] with the usual loss noted in running ordinary varnish gums such as Kauri and Zanzibar.

[1] H. A. Gardiner. Fume control in the varnish industry. Chap. II, Papers on Paint and Varnish. Washington, 1920.

The Succinic Acid Content of the Resin

A sample of 50 grams of amber was heated in a distillation flask and the products of reaction collected. The water was separated from the oil by filtration, and, on evaporating, a mass of crystals remained. These were purified by recrystallization from absolute alcohol and weighed 2·1 gm., equal to 4·2 per cent of the amber. Tests confirmed that the material was succinic acid.

Several samples of the Coalmont resin were treated similarly. The aqueous portion of the distillate had an acid reaction and gave faint indications of succinic acid, but on evaporating only a resinous material remained there not being sufficient succinic acid to determine quantitatively. The Cedar Lake resin gave similar results.

The Action of Solvents

Most fossil resins are only slightly soluble in the usual solvents, and the Coalmont and Cedar Lake resins are no exception to this.

The method used to determine solubility was as follows:

A known weight of resin powdered to pass 100 mesh screen was placed in a gooch crucible previously prepared with a mat of filter paper and weighed. The crucible was suspended in the wide neck of a special flask fitted with a reflux condenser and containing 50 c.c. solvent. The flask was placed on a sand bath and the resin subjected to the boiling solvent for several hours. The crucible was then dried and weighed, and the loss in weight calculated. The solvent was also evaporated to dryness and the residue thus obtained weighed. The two results gave fairly coincident figures.

The following are the results obtained:—

Solvent	Per cent soluble		
	Coalmont resin	Cedar Lake[1] resin	Prussian amber
	Per cent	Per cent	Per cent
Absolute alcohol	8·7	21·0	14·3
Ether	5·0	24·8	18·8
Acetone	10·8	.....	23·3
Chloroform	38·4	.....	17·3
Carbon tetrachloride	7·3	.....	11·5
Benzene	38·4	.....	21·2
Petroleum ether	5·1	.....	.....
Pyridine	51·8	.....	.....
Selenium oxychloride	partly soluble		

As in the case of amber and the varnish copals, the Canadian resins were only completely soluble in solvents after they had been melted and had lost about 25 per cent of their weight. Both varnishes were soluble in linseed oil and turpentine after being heated to 300°C., as amber is.

[1] Harrington. Amer. Jour. Sci. 1891, Vol. 42, p. 332.

Saponification Value

Samples of the Coalmont resin were boiled with 0·5 N alcoholic potash in a flask fitted with reflux condenser, and the amount of alkali required to saponify the esters present ascertained.

It was found that 1 gm. of the resin took 75 mgm. potassium hydroxide for saponification. Similar figures[1] for Prussian amber are 86·8, 74·5, 91·1 and 115.

Iodine Value

The iodine value, a measure of unsaturation of the material, was determined by Hubl's process.[2] Owing to the incomplete solubility of the resin in chloroform the results were not altogether satisfactory. The mean of several experiments gave an iodine value of 41·3 (gms. iodine required to combine with 100 gm. resin).

The corresponding value for Prussian amber[2] is 69·4.

Ultimate Analysis

Neither sulphur nor nitrogen could be detected in the Coalmont fossil resin, although the associated coal contained both elements. Sulphur was present in the Cedar Lake sample.

A trace of this substance is reported in most specimens of amber.

The following table gives the results obtained from combustion analyses of the Coalmont resin and includes figures for other varieties of fossil resins:—

Material		Carbon	Hydrogen	Oxygen	Ash	Ratio C : H
Coalmont fossil resin—						
(*a*) Air dried sample, average grade	(I)	76·40	10·19	10·66	2·75	7·17
	(II)	76·60	11·02	7·78	4·60	6·95
(*b*) Selected clear pieces, average	(I)	80·01	10·44	7·95	0·80	7·82
	(II)	79·80	10·11	9·13	0·96	7·89
Cedar lake[3] material dried over H_2SO_4 in vacuo	(I)	80·01	10·37	9·53	0·09	7·71
	(II)	79·91	10·55	9·45	0·09	7·57
Prussian amber		78·82	10·23	10·95	.....	7·71

These results show the Coalmont fossil resin and the Cedar Lake material to have a very similar composition to Prussian amber.

Analysis of the coal associated with the Coalmont resin gave the following results:—

Carbon	Hydrogen	Oxygen	Sulphur	Nitrogen	Ash
59·00	4·55	23·11	1·82	1·07	10·45 per cent

The proximate analysis was:—

Moisture	Ash	Volatile	Fixed carbon
3·48	10·45	35·06	51·01

[1] K. Dieterich. Analysis of Resins, Balsams and Gum Resins, pp. 139–140. Scott Greenwood, London, 1920.
[2] Loc. cit.
[3] Analyses by Dr. Harrington. Amer. Jour. Sci. 1891, Vol. 42, p. 332.

THE POSSIBLE INDUSTRIAL USES OF THE RESINS

For Varnish Manufacture

Several samples of varnish were made up in the laboratory according to prescribed recipes by melting samples of each of the resins, adding hot boiled linseed oil and small quantities of litharge as a drier, and thinning with spirits of turpentine. After filtering and leaving for some time these were tested and compared with some typical commercial varnishes. They gave very satisfactory results, drying in the usual time, and resulting in lustrous and hard surfaces. The chief disadvantage was the rather dark colour. Samples of the graded resins have been submitted to various varnish manufacturers, though unfortunately not in sufficient quantity for plant tests to be made. Their opinions have been without exception, that although they would undoubtedly make good varnishes, they could not compete with the large sized and much paler copals such as Zanzibar, Kauri, Manila and Congo gums. One has only to see these resins contrasted with the Canadian samples to be equally convinced. The disadvantages are their small size and their comparatively dark colour, especially evident when they are heated.

The greatest demand is for pale varnishes and the highest price is obtained for large sized, pale and clear resins. For the cheaper and dark varnishes Kauri scrapings and dust can be obtained for a few cents per lb. and much rosin is also used. It is only with these materials that the Coalmont and the Cedar Lake resins could compete, and even then it would be difficult to convince the varnish makers that the harder Canadian resins are superior.

The Oil obtained by Distillation

There is very little use for amber oil. Similarly it would be difficult to find uses for the oil obtained by distillation of the Coalmont and the Cedar Lake resins. Its property of darkening on exposure to light and air make it unsuitable for most purposes. It might be used to add to cements, artificial resins, and plastics, but its hardening and waterproofing properties are doubtful.

Amber oil is quoted at 75 cts. to $1 a pound on the New York market. Coalmont resin oil is very similar in its properties to amber oil.

Moulding Processes

Large sized blocks of amber are formed from chippings and waste by the action of heat and pressure. According to one process the material is heated to 200-250°C. and 400 atmospheres pressure is applied. Experiments made compressing Coalmont resin have not yet been successful.

However, the synthetic resins such as Bakelite, Redmanol and Condensite are rapidly supplanting compressed amber and it is improbable that there is any use for the resins in this way.

It therefore seems difficult to find satisfactory industrial uses for these materials unless they can be obtained in considerable quantity and sold cheaply enough to compete with the darker varnish resins.

II

A FIELD METHOD AND APPARATUS FOR THE DETERMINATION BY MEANS OF ELECTRICAL CONDUCTIVITY MEASUREMENTS, OF THE CHARACTER OF WATERS LEAKING INTO OIL AND GAS WELLS.

R. T. Elworthy

The development of the apparatus, its calibration and preliminary testing as hereunder described were carried out for and at the request of the Northwest Territories and Yukon branch of the Department of the Interior. One of the functions of that branch is the supervision and inspection of oil and gas wells in the western provinces, especially in relation to drilling operations. An important part of this work is the safeguarding and protection of the oil and gas sands from encroachment by water. Drillings in progress must be prevented from flooding. Such water may come from the lower levels of the oil or gas bearing formation, when it is usually known as bottom water, or it may infiltrate from one or more of the water bearing strata that have been penetrated in boring. In the latter case the water is more probable to be a lightly mineralized water, originally a surface water which has gained soluble constituents in its passage through the earth. Although mixtures of bottom waters and top waters are often met with, it can be generally stated that they are distinctly different in character, in that the bottom waters are much more strongly mineralized, having sodium chloride as the predominating constituent. They are usually considered to be remnants of the original sea in which the sediments were laid down. Greater difficulty exists, however, in distinguishing between various top waters.

It has been proposed that samples of waters from various known horizons and from representative wells in any particular oil or gas field be collected and chemically analyzed. From such data information can be afterwards obtained of the source of water leaking into new or unexamined wells in that field, when samples of these waters are analyzed.

Valuable investigations of this nature have been carried out in the United States.[1]

It was proposed that such a survey should be made in the various gas fields of Alberta, but it was not possible to carry out this investigation on account of the lack of facilities and of staff. It was therefore thought that a determination of the electrical conductivity of the waters would give much information in less time. This examination would be made at first on a series of waters whose origin and general character was known, and then using the measurements obtained as standards. it would be applied to waters of unknown character and origin encountered in new drillings.

This report describes the apparatus that was chosen for the purpose and some of the preliminary results obtained.

[1] G. S. Rogers. "The chemical relations of the oil field waters in the San Joaquin Valley." U.S. Geol. Survey, Bull. 653, 1917. R. O. Neal. "Petroleum Hydrology applied to the Mid Continent field." Bull. Amer. Inst. Min. Met. Bull. No. 145, pp. 1-8, 1919 and No. 147, pp. 603.

Electrical Conductivity Measurements in Water Analysis

There has been a considerable development of the use of conductivity measurements in industrial processes[1] during the last few years. Although the physical laws underlying the conduction of electricity by concentrated aqueous solutions of salts are still subjects of scientific research, the methods employed can be made to give results far above the accuracy required in industrial operations. Such measurements have been used in checking the composition of boiler feed water, in following electrolytic separations of metals, in the washing of wood pulp in the sulphate process, and in many similar operations. In water analysis the method has been applied to determine[2] the mutual influence of various salts on their dissociation in aqueous solution, and for the study of the relation between salt content of natural waters and conductivity.[3] Similar measurements afford a rapid means of determining the total solids in a water sample.

The results of analysis of several waters typical of those met with in the western gas and oil fields showed that sodium chloride was the main constituent, and therefore the method resolved itself into one that would best give by simple manipulation the conductivity of solutions of sodium chloride.

Analysis of Typical Waters from Alberta Gas Fields

The following table gives the results of analysis of five waters, the first of a series to be collected as typical of those encountered in drilling for oil and gas in Alberta.

(*For Table—see page 60.*)

The following particulars of the geological relations of the strata from which the samples came were supplied by Mr. S. E. Slipper, petroleum engineer of the Northwest Territories and Yukon Branch.

No. 1.—The water horizon was 55 feet below the Cretaceous and thought to be in the upper Devonian. Fresh waters occurred at 224 feet, 440 feet and 500 feet, but no leakage from there was reported. This is confirmed by the analysis which shows the water to be strongly mineralized. It is probably a connate or fossil sea water.

No. 2.—This water is probably a mixture of meteoric and connate water, although it is believed to issue from an horizon 150 feet deep in the Cretaceous formation.

No. 3.—The source of this sample was from the Ribstone Creek beds of the Belly River formation, 350-430 feet in depth.

No. 4.—This water comes from a series of sandstone beds below the Loon River shales. It is highly charged with natural gas and occasional globules of heavy oil come up with the water. This water occurs stratigraphically higher than No. 5.

No. 5.—This sample issues from several sandstone beds in the base of the Cretaceous. It is ejected from the wells with considerable force and often forms geysers rising to forty feet in the air. Large quantities of gas occur with the water. When closed, the pressure varies from 230 to 180

[1] E. A. Keeler, Chem. and Met. Eng. Vol. 23, p. 721, 1920.
[2] Chem. Weekblad 15, pp. 1160-83, 1918.
Chem. Abs. 13, p. 624, 1919.
[3] Chem. Weekblad 8, pp. 977-82.

Sources	Czar Monitor Misty Hills field. Tit Hills well No. 1		Czar Monitor Misty Hills field. North West Oil and Exploration Co., No. 1		Birch Lake field. Birch Lake well No. 1		Peace River field. Peace River Petroleum Co.'s well No. 1		Peace River field. Peace River Oil Co. well No. 3	
Number	1		2		3		4		5	
Depth of water, horizon ft	3,295		150		350–430		1137–1148		1003–1282	
Specific gravity at 15° C	1·052		1·004		1·001		1·004		1·020	
Constituents	P.P.M.	R.V.%	P.P.M.	R.V.%	P.P.M.	R.V.%	P.P.M.	R.V.%	P.P.M.	R.V.%
Carbonic acid (CO_3)	.....	.....	.....	.....	.....	.....	.....	.....	257·8	1·02
Bicarbonic acid (HCO_3)	324·0	0·2	693·0	27·98	481·0	7·9	2,051·0	5·1	3,331·0	6·48
Sulphuric acid (SO_4)	13·0	.....	78·7	4·02	17·0	0·4	9·0	.....	127·8	0·30
Chlorine (Cl)	42,395·0	49·8	258·9	18·00	1,470·0	41·7	10·355·0	44·9	12,549·0	42·20
Sodium (Na)	24,943·0	45·4	414·8	44·43	1,033·0	45·2	6,829·0	45·6	9,363·0	48·51
Potassium (K)	883·0	0·9	31·4	1·97	75·0	1·9	21·0	0·1	168·0	0·50
Calcium (Ca)	898·0	1·8	19·9	2·44	35·0	1·8	189·0	1·4	28·8	0·18
Magnesium (Mg)	580·0	1·9	5·8	1·16	13·5	1·1	230·0	2·9	82·8	0·81
Iron oxide and alumina	85·5	.....	5·9	.....	38·0	.....	17·7	.....	17·6	.....
Silica (SiO_2)	0·9	.....	5·9	.....	35·7	.....	11·2	.....	44·1	.....
Total	70,122·4	100·0	1,514·3	100·0	3,198·2	100·0	19,712·9	100·0	25,965·0	100·0
Hypothetical combinations—	P.P.M.	%	P.P.M.	%	P.P.M.	%	P.P.M.	%	P.P.M.	%
Sodium chloride (NaCl)	63,455·0	90·7	380·0	25·1	2,311·0	72·4	17,043·0	85·6	20,440·0	78·8
Potassium chloride (KCl)	1,684·0	2·4	60·3	4·0	143·0	4·4	37·2	0·2	313·0	1·2
Sodium sulphate (Na_2SO_4)	.....	.....	115·6	7·6	24·8	0·7	14·2	0·1	184·5	0·7
Magnesium chloride ($NaCl_2$)	2,270·0	3·2	.....	.....	.....	.....	.....	.....	.....	.....
Magnesium sulphate ($MgSO_4$)	20·0	.....	.....	.....	.....	.....	.....	.....	.....	.....
Sodium carbonate (Na_2SO_3)	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Sodium bicarbonate ($NaHCO_3$)	421·0	0·5	832·0	54·9	422·5	13·3	445·0	2·2	16·0	0·3
Magnesium bicarbonate Mg (HCO_2)	.....	.....	34·4	2·2	81·5	2·5	1,375·0	6·9	4,589·0	17·6
Calcium bicarbonate Ca (HCO_3)	.....	.....	80·2	5·3	141·7	4·4	769·0	3·8	286·0	7·1
Calcium chloride ($CaCl_2$)	2,186·0	3·1	.....	.....	.....	.....	.....	.....	75·0	0·3
Ferric oxide and alumina	85·5	0·1	5·9	0·4	38·0	1·1	17·7	0·1	.....	.....
Silica (SiO_2)	0·9	.....	5·9	0·4	35·7	1·1	11·2	0·1	17·6	.....
Total	70,122·4	100·0	1,514·3	100·0	3,198·2	100·0	19,712·9	100·0	44·1	0·2
Properties of reaction in per cent—									25,965·2	100·0
Primary salinity	92·6		44·04		84·2		89·8		85·00	
Secondary salinity	7·0									
Primary alkalinity			48·76		86·2		1·6		13·02	
Secondary alkalinity	0·4		7·20		9·6		8·6		1·98	

pounds per square inch, but is probably a gas pressure over and above the pressure of the water column in the well. There is a distinct odour of hydrogen sulphide from the flow of water and gas, and the water deposits a coating on material over which it flows. Heavy oil and tar sand beds are interlayed with the water sands in the well. The sandstones are overlain by impervious shales and beneath them are the Palæozoic limestones.

Conductivity Measurements

It will be seen from the results of analyses of these waters that the chief constituent is sodium chloride, usually forming more than 75 per cent of the total solids. Preliminary conductivity measurements made with the usual set up of a Wheatstone's bridge, induction coil and telephone gave encouraging results.

For such measurements to be carried out in the field, a reliable and simple form of apparatus had to be selected.

Three types of instrument were therefore tested.

The first was an instrument, made by Messrs. Leeds and Northrup of Philadelphia, designed especially for measurements of conductivity in industrial processes. This apparatus gave very satisfactory results in many ways, and was well built and rugged in construction; but it required an alternating current of 110 volts to operate it. This rendered it of no value for field use.

The second instrument was the Dionic water tester, made by Messrs. Evershed and Vignoles, of Chiswick, London. It is intended for the detection and measurement of impurities in town water supplies, boiler waters, river waters, and washing waters in all types of industrial processes. The electric current is generated by a small dynamo worked by hand, and the conductivity of a sample of water, contained in a special cell, is read off directly on the scale of a galvanometer of the pointer type. The instrument at our disposal however was intended for very lightly mineralized waters, and even when a cell with greater distance between the electrodes was substituted, it did not give a sufficiently wide range of readings. Otherwise the apparatus would be ideal for field use.

The third instrument, and the one finally adopted, was the conductivity bridge[1] designed for the measurement of soluble salts in soils by the Bureau of Soils, U.S. Department of Agriculture. It was purchased from the Bryan Instrument Co., Washington.

This is of the regular Wheatstone bridge type, and measurements are made by adjusting the position of a slider on a resistance wire till there is a minimum of sound in the telephone.

The conductivity cell supplied with the instrument was unsuitable for testing strong saline solutions and a more suitable cell was therefore designed.

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[1] U.S. Dept. of Agriculture. Bureau of Soils, Bull. No. 61, 1910. The electrical bridge for the determination of soluble salts in soils, by R. O. E. Davis and H. Bryan.

Conductivity Cell

Fig. 2 and Plate I show the design of this cell. It consists of a glass cylinder, 1¼ inch in diameter, and 7 inches long, closed at the top with a rubber stopper carrying the supports for the two electrodes. These are circular platinum discs, 9 mm. diameter, welded to platinum wires sealed into glass tubes serving as supports, and carrying the copper wires making connection with the bridge.

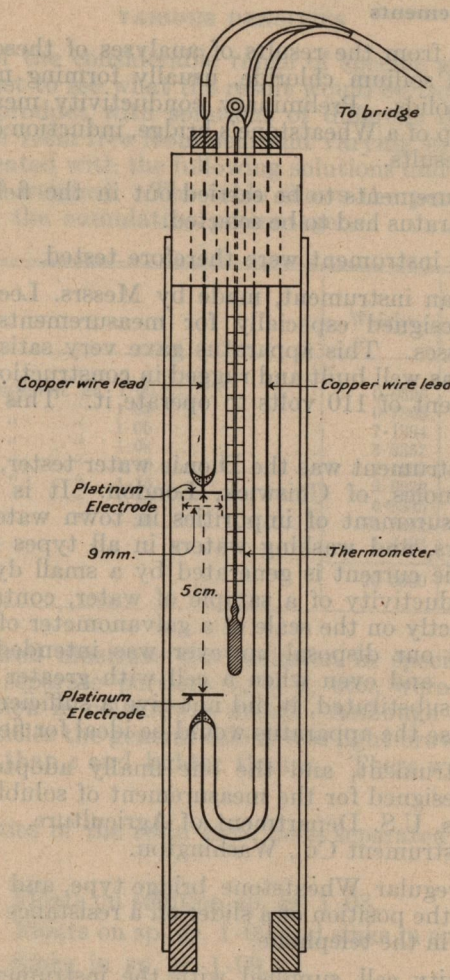
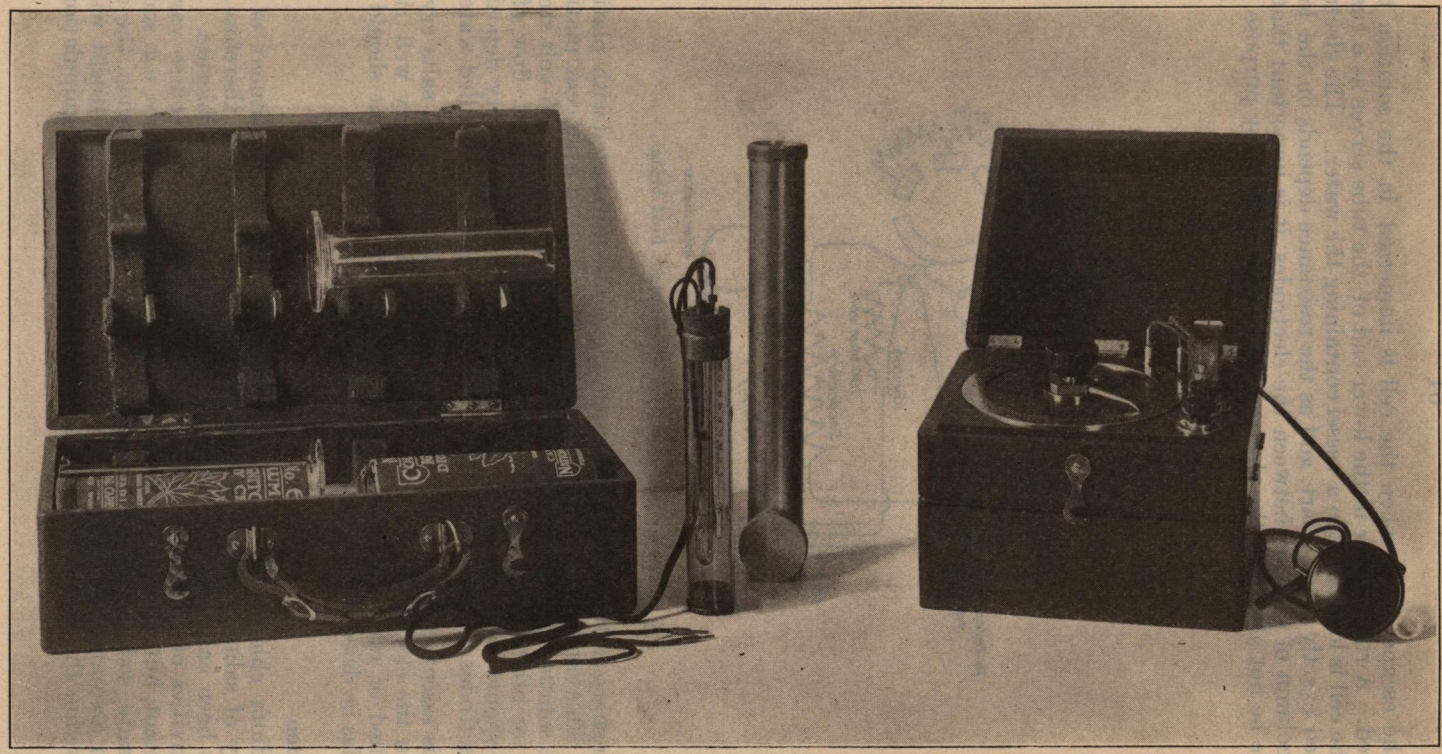


FIG. 2. Conductivity cell.

Fig. 3 shows the wiring of the circuits (see Bull. 61, pp. 10-13). The temperature of the solution at which the measurements are made is read on the thermometer in the cell. Holes in the glass tube and the stopper



Conductivity bridge, cell and carrying case.

allow the escape of air when the cell is immersed in the solution to be measured. A rubber ring in the lower end of the tube serves as a cushion when the cell is lowered into a vessel containing the water. The electrodes are fitted exactly 5 cm. apart, and as the resistance depends on the length of the column of solution between the electrodes, it is important that the distance be not altered if the electrodes are removed for any purpose.

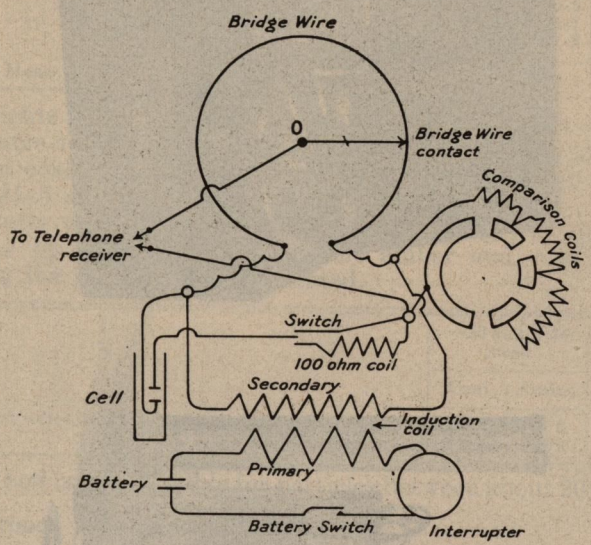


FIG. 3. Diagram of interior connections for field bridge.

For sharp setting of the balance the electrodes are coated with platinum black. This can be easily done by putting the cell into a 5 per cent platinum chloride solution and connecting the leads from the cell to a 6-volt storage battery or other source of direct current. The direction of flow of the current should be reversed every minute until a black velvety deposit on both electrodes is obtained. The cell is then washed in distilled water.

It is essential that the cell be kept clean and free from salts crystallizing on the electrodes or walls. After use it should be very well rinsed in distilled or rain water and left to drain. On no account should the electrodes be dried with a cloth, or even touched.

Calibration

Sodium chloride is by far the most important constituent of the majority of saline waters found in oil and gas wells, and, therefore, the cell has been calibrated by means of solutions of sodium chloride. The conductivities of equivalent solutions of different salts of course vary to some extent, but with such salts as are found in mineral waters, e.g. sodium bicarbonate, sodium sulphate, magnesium and calcium chloride, the error made in assuming the sodium chloride to be the sole constituent will be easily within the limits of accuracy of measurement and manipulation.

Temperature Effect

The electrical conductivity of a solution has a large temperature coefficient, being approximately 2 per cent per degree centigrade. This is greater than the errors of measurement, and therefore must be allowed for in the calculations. The actual value for a 5 per cent sodium chloride solution at 18°C. is 0·0198.[1] Measurements made at three different temperatures, 20°, 25° and 30°C., recorded in the table following, give approximately 2 per cent difference per degree.

The conductivity of a solution increases with increase of temperature, and vice versa, and, therefore, the resistance which is the reciprocal of conductivity, decreases with increase in temperature. For example, a salt solution at 20° had a resistance of 103, at 25° it was 94, and at 30°, 87. To bring the results to 25°C. therefore, the resistance found at temperatures above 25° must be increased by 2 per cent per degree, and those measured below 25° must be reduced by 2 per cent per degree.

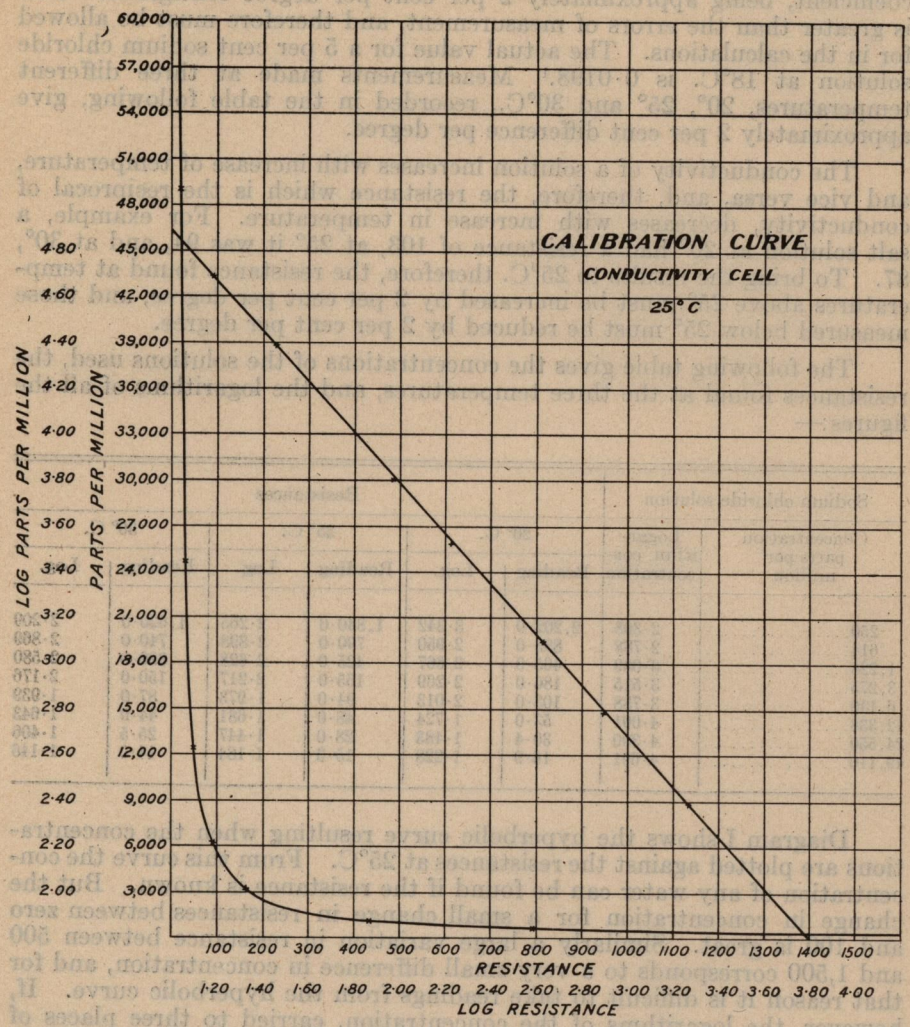
The following table gives the concentrations of the solutions used, the resistances found at the three temperatures, and the logarithms of all the figures:—

Sodium chloride solution		Resistances					
Concentration parts per million	Logarithm concentration	20° C.		25° C.		30° C.	
		Reading	Log.	Reading	Log.	Reading	Log.
250	2·398	2,202·0	3·342	1,840·0	2·265	1,620·0	2·209
614	2·788	893·0	2·950	790·0	2·898	740·0	2·869
1,228	3·089	465·0	2·667	425·0	2·628	380·0	2·580
3,275	3·515	186·0	2·269	165·0	2·217	150·0	2·176
6,140	3·788	103·0	2·013	94·0	1·973	87·0	1·939
12,338	4·091	53·0	1·724	48·0	1·681	44·0	1·643
24,550	4·390	30·4	1·483	28·0	1·447	25·5	1·406
49,110	4·691	16·9	1·228	15·3	1·184	14·0	1·146

Diagram I shows the hyperbolic curve resulting when the concentrations are plotted against the resistances at 25°C. From this curve the concentration of any water can be found if the resistance is known. But the change in concentration for a small change in resistances between zero and 100 is great. Similarly a large variation in resistance between 500 and 1,500 corresponds to a very small difference in concentration, and for that reason it is difficult to take readings from the hyperbolic curve. If, however, the logarithms of the concentration, carried to three places of decimals, be plotted against the logarithms of the corresponding resistances, a straight line results when the points are joined, also shown in Diagram I. It is much easier to take readings from this straight line, and for use in the field the logarithmic curves for the temperatures 20°, 25° and 30°C. were plotted.

[1] Landolt, Bornstein and Roth. Physikalische Tabellen, Berlin, 4th edition, p. 1092.

DIAGRAM I


CALIBRATION CURVE
CONDUCTIVITY CELL
25° C
LOG PARTS PER MILLION
PARTS PER MILLION
4·80 4·60 4·40 4·20 4·00 3·80 3·60 3·40 3·20 3·00 2·80 2·60 2·40 2·20 2·00
60,000 57,000 54,000 51,000 48,000 45,000 42,000 39,000 36,000 33,000 30,000 27,000 24,000 21,000 18,000 15,000 12,000 9,000 6,000 3,000
100 200 300 400 500 600 700 800 900 1000 1100 1200 1300 1400 1500
RESISTANCE
1·20 1·40 1·60 1·80 2·00 2·20 2·40 2·60 2·80 3·00 3·20 3·40 3·60 3·80 4·00
LOG RESISTANCE

Application.—The five samples of water, the analyses of which have just been given, were tested by the conductivity method. Taking the resistances found as recorded in column 2 and using Diagrams II and III the numbers in column 4, which are the logarithms of the corresponding concentrations, were obtained. The antilogarithms, or the actual concentrations (parts per million), found from tables, are in the fifth column. The sixth column gives the results found several months ago by actual evaporation, drying and weighing:—

Sample No.	Resistance found at 25° C.	Logarithm resistance	Logarithm corresponding concentration from curves.	Concentration looking up figs. of last column in antilog. tables	Concentration found by actual analysis
I	10·8	1·033	4·85	70,790	68,080
II	432·0	2·635	3·10	1,259	955
III	224·0	2·350	3·42	2,630	2,371
IV	34·5	1·538	4·29	19,500	19,360
V	28·0	1·447	2·39	24,550	25,120

DIAGRAM II

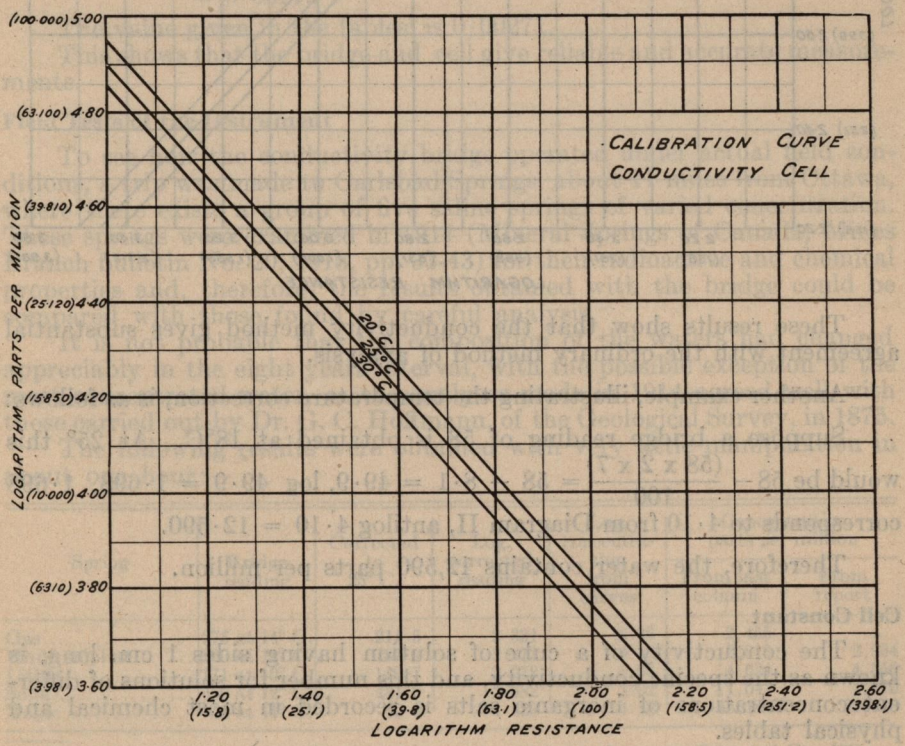
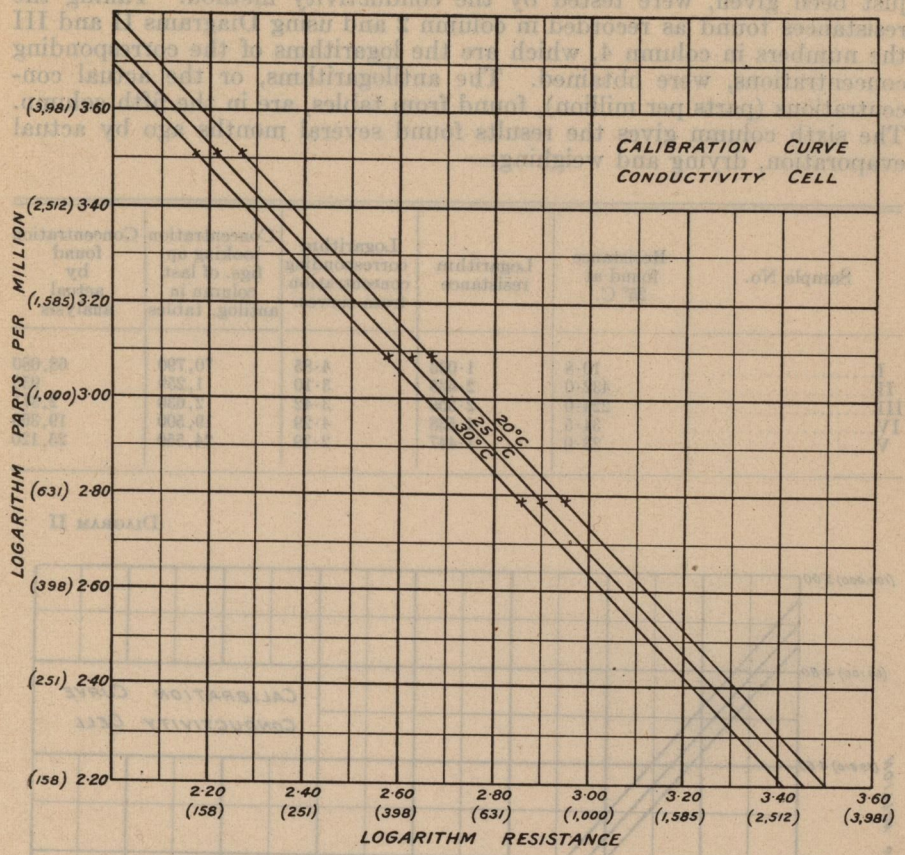


DIAGRAM III



These results show that the conductivity method gives substantial agreement with the ordinary method of analysis.

Another example, illustrating the temperature correction, is as follows:

Suppose a bridge reading of 58 is obtained at 18°C. At 25° this would be $58 - \frac{(58 \times 2 \times 7)}{100} = 58 - 8{\cdot}1 = 49{\cdot}9$, log $49{\cdot}9 = 1{\cdot}698$, $1{\cdot}698$ corresponds to $4{\cdot}10$ from Diagram II, antilog $4{\cdot}10 = 12{\cdot}590$.

Therefore, the water contains 12,590 parts per million.

Cell Constant

The conductivity of a cube of solution having sides 1 cm. long, is known as the specific conductivity, and this number for solutions of different concentrations of inorganic salts is recorded in most chemical and physical tables.

The usual type of conductivity cell on account of its large dimensions gives a number greater than the specific conductivity, but the following relation holds:—

Specific conductivity (c) = conductivity (C) x cell constant (K). To find the cell constant of the conductivity cell described in this report, the following measurements were carried out:—

Two solutions of potassium chloride were made up, one 0·1N. (7·46 grams per litre) and the other, 0·02 N. (1·492 grams per litre). The 0·1N. solution gave a reading on the bridge at 24°C. of 91·5 ohms; the 0·02 solution gave 420. This corresponds to a conductivity of

$$\frac{1}{91\cdot5} = 0\cdot01092 \text{ and of } \frac{1}{420} = 0\cdot00238.$$

The specific conductivity of 0·1N potassium chloride solution at 24°C. is 0·01264.[1] Applying the relation, we have

$$0\cdot01264 = 0\cdot01092 \times K$$

or $$K = 1\cdot155$$

Taking this value and the conductivity found in the case of the 0·02N potassium chloride solution, the specific conductivity of this solution at 24°C. is:—

$$c = 1\cdot155 \times 0\cdot00238$$
$$= 0\cdot00275$$

The value given in the tables[1] is 0·00271.

This shows that the bridge and cell give reliable and accurate measurements.

Field Tests of the Instrument

To see how the conductivity bridge operated under actual field conditions, a trip was made to Carlsbad Springs, about 11 miles from Ottawa, where there exists a group of five saline springs of varied concentration. These springs were examined in 1914 (Mineral Springs of Canada, Mines Branch bulletin No. 20, 1918, pp. 33-43) for their radioactive and chemical properties and, therefore, the results obtained with the bridge could be compared with those found by careful analysis.

It is not probable that the composition of the waters had changed appreciably in the eight years interval, with the possible exception of the most concentrated water, as the analyses made in 1914 agreed well with those carried out by Dr. G. C. Hoffmann, of the Geological Survey, in 1875.

The following results were obtained with very little manipulation in about one hour:—

Spring	Bridge reading	Corrected to 25° C.	Log. corrected reading	Log. concentration from curves	Concentration parts per million	
					From last column	From report
Gas	275 at 14° C.	214·5	2·331	3·39	2,455	.....
White sulphur	260 at 11° C.	187·2	2·272	3·45	2,818	2,964
Lithia	171 at 10° C.	119·7	2·078	3·66	4,624	4,550
Soda	650 at 12° C.	481·0	2·682	3·02	1,047	1,170
Magic	59 at 11° C.	42·5	1·628	4·18	15,140	22,140

[1] Physical and chemical constants, Kaye and Laby, p. 86, published by Longmans, Green and Co., 1919.

Considering that the instrument was calibrated with sodium chloride solutions, and that considerable temperature corrections have been made, these results are very satisfactory. The discrepancy in the case of the Magic Spring may be perhaps due to a decline in concentration, especially as this was found to have occurred between 1875 and 1914.

The waters were afterwards tested in the laboratory at 25°C. with the following results:—

—	Gas	White sulphur	Lithia	Soda	Magic
Found	210·0	190·0	120·0	520	43·0
Calculated	214·5	187·5	119·7	481	42·3

The percentage of sodium chloride of the total inorganic matter in solution in these waters was, according to the analyses in 1914:—

—	White sulphur	Lithia	Soda	Magic
Sodium chloride (NaCl) per cent	69·30	77·11	44·05	73·50
Sodium bicarbonate ($NaHCO_3$) per cent	21·49	9·51	48·40	.....

The instrument has now been sent out for field use in western Canada, where it is being tested on a number of waters from known horizons, so that a series of standards can be set up.

The determination of total solids by this means seems to afford a method sufficiently accurate for the purpose required.

ORE DRESSING AND METALLURGICAL DIVISION

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm, *Chief of Division*

During the year 1922, the officers of the Ore Dressing and Metallurgical Division were engaged on investigations in connection with the treatment of Canadian ores, covering a wide variety of ore dressing and metallurgical problems. Some of these investigations were a continuation of the experimental work of the former year, and others were new investigations undertaken during the year The extent and scope of the experimental work can be seen from the table compiled of the ores and metallurgical products received for investigation. A brief review of the investigations is given below.

Test No. 128

THE RECOVERY OF PLATINUM IN BLACK SANDS, BY THE ZACHERT PROCESS, BY R. K. CARNOCHAN

Tests were made on the recovery of the platinum in the concentrates from the Bullion Mine, Quesnel district, B.C., by the use of the Zachert process. The results of the tests conducted were not favourable. By ordinary methods of amalgamation as good a recovery could be expected.

Test No. 156

THE RECOVERY OF THE VALUES IN A SILVER-LEAD ORE FROM ARROWHEAD, B.C., BY R. K. CARNOCHAN

This was a silver-lead ore carrying gold as its chief valuable mineral. Of the various methods tried that of table concentration followed by cyanidation gave the best results.

Test No. 161

THE CONCENTRATION OF THE FLIN FLON DISSEMINATED ORE, BY C. S. PARSONS

During the previous year, an investigation was conducted on the treatment of the sulphide ore of the Flin Flon ore body. During 1922 experimental work was conducted on the disseminated ore from the footwall and hanging wall portions of this ore body. The presence of a deleterious gangue mineral made the concentration of this ore a more difficult problem than first supposed from an examination of the ore. Methods of overcoming the difficulties are described in the report.

Test No. 162

THE ELIMINATION OF THE IMPURITIES FROM MALAGASH ROCK SALT, BY R. K. CARNOCHAN

This investigation covers the experimental work on the elimination of impurities in the Malagash rock salt. Due to the presence of iron and organic matter in the salt the products for the trade were slightly discoloured. Before embarking on an expensive process of dissolution and evaporating, other methods of purification were tried.

Test No. 163

THE CONCENTRATION OF THE GRAPHITE ORE FROM THE TIMMINS MINE, WESTPORT, ONT., BY C. S. PARSONS

The experimental work was conducted to determine the best method of producing a high grade flake with a maximum recovery of coarse flake and a high recovery of carbon values in the ore.

Test No. 164

SEPARATION OF FOSSIL RESIN FROM COAL, BY R. K. CARNOCHAN

This investigation covers the experimental work on the separation of fossil resin from the coal of the Coalmont Collieries, Ltd., Coalmont, B.C. Methods of recovering the resin from the coal are described in the report.

Test No. 165

THE WET SEPARATION OF ASBESTOS FROM ITS GANGUE, BY R. K. CARNOCHAN

The results of this preliminary investigation show the possibility of the application of the wet method of milling asbestos rock for the recovery of the fibre. If such a process could be demonstrated to be feasible at a reduced cost over the dry methods in use, a decided advance in the practice of milling asbestos rock would be made.

Test No. 166

THE SEPARATION OF FLUORITE, CALCITE, AND BARITE IN THE FLUORSPAR FROM THE MADOC DISTRICT, ONT., BY R. K. CARNOCHAN

The experimental work conducted on this type of fluorspar has revealed a new method of separation applicable to the separation of the minerals of these deposits. By this new method described in the report, an exceptionally high grade fluorspar can be obtained.

Test No. 167

THE MILLING AND CONCENTRATION OF COPPER-GOLD ORE FROM KITSALAS MOUNTAIN COPPER CO., USK, B.C., BY C. L. DEWAR

The report of this investigation shows that a very high grade copper concentrate 50 per cent–55 per cent copper with recoveries of 90 per cent of the copper, gold, and silver values can be obtained from this ore. The ore is very amenable to the method described in the report.

Test No. 168

THE CONCENTRATION OF THE ANTIMONY ORE FROM LAKE GEORGE, N.B., BY C. S. PARSONS

The report of this investigation covers the most practicable method for the recovery of the antimony values in a high grade concentrate with a high recovery. From an ore assaying 11·65 per cent antimony a product is obtained assaying 60 per cent–65 per cent antimony with a recovery of 95 per cent of the antimony values. This was demonstrated by a tonnage check test confirming the results of the small scale tests.

Test No. 169

THE RECOVERY OF THE SILVER VALUES IN THE LEACHED CHLORIDIZED RESIDUES OF THE DOMINION REDUCTION CO., COBALT, ONT., BY C. S. PARSONS AND C. L. DEWAR

The experimental work on these residues shows that of the various methods tried for the recovery of the silver values, the one which gave the best results was an acid wash, and cyanidation of the washed residues. By this method, 75 per cent to 80 per cent of the silver values were extracted with a consumption of cyanide of about 10 pounds per ton of residues.

Test No. 170

GOLD ORE FROM BEAR RIVER, BEDWELL SOUND, VANCOUVER IS., B.C., BY R. K. CARNOCHAN

This investigation covers the methods employed in the treatment of a gold ore carrying metallic sulphides and arsenides of copper, iron, lead, and zinc. Conclusions are drawn as to those methods most applicable to this class of ore.

Test No. 171

THE CONCENTRATION OF THE RADIO ACTIVE MINERALS IN A SHIPMENT OF PEGMATITE FROM KEARNEY, ONT., BY R. K. CARNOCHAN

The report of this investigation shows that very small amounts of radio active minerals are present in this pegmatite, and that it could not be classed as a source of radium.

Test No. 172

THE PRECIOUS METALS AND OTHER VALUES IN THE COPPER-NICKEL ORES OF SHEBANDOWAN LAKE, ONT., BY H. C. MABEE

This is a report of analysis of five samples submitted by Dr. T. L. Tanton of the Geological Survey on the presence of a considerable quantity of the metals of the platinum group, especially palladium, in the copper-nickel-cobalt ores of Shebandowan lake west of Port Arthur, Ont.

Test No. 173

THE RECOVERY OF THE VALUES FROM A RICH SHIPMENT OF GOLD ORE FROM THE CARIBOU MINING DISTRICT, N.S., BY R. K. CARNOCHAN

This is a report on the methods employed in recovering the values from a gold ore, rich in metallics. Other than showing the spotty nature and characteristics of Nova Scotia gold ores, little of value in treating Nova Scotia gold ores can be obtained from the report.

Test No. 174

THE RECOVERY OF THE VALUES FROM THE ORE OF THE E. H. GLADWIN MINE, BEAVER DAM MINING DISTRICT, N.S., BY R. K. CARNOCHAN

This report is a description of the experimental work with the results obtained, on the ore from this mine, with conclusions as to the best methods for the recovery of the values.

Test No. 175

AN INVESTIGATION OF THE TAILING AND ROCK DUMPS AT THE GOLD MINES OF NOVA SCOTIA, WITH A REVIEW OF THE PAST MINING AND MILLING METHODS, THE PRESENT STATUS OF THE GOLD INDUSTRY IN THE PROVINCE, AND SOME SUGGESTIONS FOR THE REVIVAL OF THE INDUSTRY, BY C.S. PARSONS.

This report is the result of a field and laboratory investigation by Mr. C. S. Parsons, conducted during the latter part of the year. Due to the interest being taken in the development of the gold industry of the country, it was thought advisable to determine the possibility of treating the old tailing and rock dumps of the Nova Scotia gold mines, if they contained sufficient values, and at the same time revive interest in the re-opening of the more promising gold mines of the province. Very few of the dumps were found to contain sufficient values to be worked at a profit.

Test No. 176

CONCENTRATION TESTS ON TAILING SAMPLES FROM TWO MILLS OPERATING ON TYPICAL NOVA SCOTIA GOLD ORES, BY C. S. PARSONS AND R. K. CARNOCHAN

These tests were conducted to determine in what state the gold was present in the tailings from the two operating mills; whether it was free and could be amalgamated, or locked up in the sulphides. It was found that in both cases the mills were doing efficient amalgamation, and the gold in the tailings would have to be recovered by other means.

Test No. 177

THE SEPARATION OF DOLOMITE FROM MAGNESITE IN THE GRENVILLE MAGNESITES, BY R. K. CARNOCHAN

This is a report on some further experimental work on the separation of dolomite from magnesite by calcining, slacking, and classification of the slacked calcined product by washing and other means. The best magnesite product obtained contained 7 per cent lime.

Test No. 178

THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE ORE OF THE STIRLING MINE, RICHMOND CO., CAPE BRETON, BY C. S. PARSONS AND R. J. TRAILL

The report of this investigation shows a method of concentration to obtain a marketable zinc product. Leaching tests showing the solubility of the zinc and copper in weak acid solution are also given.

Test No. 179

CONCENTRATION TESTS ON LE ROI NO. 2 DUMP ORE, BY C. L. DEWAR

This report covers the tests conducted on the concentration of this ore by tabling and flotation. The tests were made mainly to determine the results of using various flotation reagents on the concentration of Rossland copper-gold ores.

Test No. 180

THE USE OF FLOTATION REAGENTS MANUFACTURED IN CANADA, BY C. S. PARSONS AND C. L. DEWAR

This investigation was undertaken at the request of Canadian manufacturers of flotation reagents in an endeavour to stimulate the use of Canadian products instead of imported ones. Tests were made on Canadian ores on which flotation was employed for the concentration and separation of the minerals, and a comparison of results was attempted using Canadian manufactured reagents as against the reagents in use. In many cases it was found that reagents manufactured in Canada could be used with equal results, but the chief difficulty lay in introducing a new reagent for one which had already given satisfaction and with which the operators were familiar.

Tests Nos. 181 and 182

THE METALLURGICAL TREATMENT OF CANADIAN PYRRHOTITE ORES BY LEACHING AND ELECTROLYTIC DEPOSITION AND OF CANADIAN PYRITE ORES BY ROASTING, LEACHING, AND ELECTRO-DEPOSITION, WITH THE PRODUCTION OF ELECTROLYTIC IRON AND RECOVERY OF BY-PRODUCTS SUCH AS COPPER, SULPHUR, ETC., BY R. J. TRAILL

The report on this investigation covers the general features of the Eustis and Estelle processes, and the preliminary experimental work on the Smith mine pyrrhotite from the west side of Lake Memphremagog, Que. (Test 181) and on the pyrite from the Eustis Mine, Que. (Test No. 182). After a preliminary study of the research carried out to date and the reports of engineers on the processes, the possibilities of the application of such processes to the large deposits of Canadian pyrrhotites and pyrites were recognized.

Tests Nos. 183 and 184

THE PRECIOUS METAL VALUES IN THE CONCENTRATION PRODUCTS OF THE NICKEL-COPPER ORES OF THE SUDBURY DISTRICT, BY H. C. MABEE

Test No. 183 is a report on the precious metal values in the concentration products from a low grade ore from the Murray mine. Test No. 184 is a report on the precious metal values in the concentration products from the lower grade ore from the No. 3 (Frood) mine. Both these reports show that the precious metal values are concentrated with the nickel and copper in the flotation of these ores.

II

LIST OF ORES AND METALLURGICAL PRODUCTS ON WHICH EXPERIMENTAL TEST AND RESEARCH WORK WAS CONDUCTED

W. B. Timm

In the following table is given a list of the ores and metallurgical products received, with test number, class of ore or metallurgical product, source of shipment, shipper, and weight of shipment, on which experimental test and research work was conducted:—

Test No.	Ore or product	Source of shipment	Shipper and address	Weight pounds
128	Platinum sands	Bullion mine, B.C.	Geological Survey	465
156	Silver-lead	Arrowhead, B.C.	Mansfield Mining Co., Mansfield, Wash., U.S.A.	220
161	Copper	Flin Flon mine	Mining Corporation of Canada, Ltd., Toronto, Ont.	360
162	Rock salt	Malagash, N.S.	Chambers and MacKay, New Glasgow, N.S.	900
163	Graphite	Timmins mine, Westport, Ont.	H. P. H. Brumell, Buckingham, Que.	20
164	Fossil resin	Coalmont, B.C.	Coalmont Collieries, Coalmont, B.C.	400
165	Asbestos	Black Lake, Que.	Black Lake Asbestos and Chrome Co., Black Lake, Que.	200
166	Fluorite	Wallbridge mine, Madoc, Ont.	Gavin M. Wallbridge, Madoc, Ont.	2,000
167	Copper-gold	Kitsalas Mt.	Kitsalas Mountain Copper Co., Usk, B.C.	100
168	Antimony	Lake George, N.B.	North American Antimony Smelting Co., Lake George, N.B.	2,154
169	Silver	Cobalt, Ont.	Dominion Reduction Co., Cobalt, Ont.	1,500
170	Gold	Bear River, Bedwell Sound, Vancouver Is.	J. B. Woodworth, Vancouver, B.C.	200
171	Radium	Kearney, Ont.	Ryan and Mann, Kearney, Ont.	19,019
172	Nickel-copper	Shebandowan lake, Ont.	Dr. T. L. Tanton, Geological Survey	100
173	Gold	Herman Hall mine, Caribou, N.S.	W. R. Hitchcock, Cornwall, Ont.	90
174	Gold	E. H. Gladwin mine, Beaver Dam, N.S.	W. R. Hitchcock, Cornwall, Ont.	575
175	Gold	Nova Scotia	C. S. Parsons, Mines Branch	4,000
176	Gold	Nova Scotia	C. S. Parsons, Mines Branch	20
177	Magnesite	Grenville Co., Que.	North American Magnesite Co., Calumet Que.	30,000
178	Zinc	Stirling mine, Cape Breton Is.	F. M. Connell, Toronto, Ont.	400
179	Copper-gold	Le Roi No. 2 mine, Rossland, B.C.	Douglas Lay, Rossland, B.C.	100
180	General investigation on flotation reagents—			
181	Iron (pyrrhotite)	Smith mine, Lake Memphremagog, Que.	F. A. Eustis, Boston, Mass.	400
182	Iron (pyrite)	Eustis mine, Eustis, Que.	F. A. Eustis, Boston, Mass.	400
183	Nickel-copper	Murray mine, Nickelton, Ont.	British America Nickel Corp., Nickelton, Ont.	200
184	Nickel-copper	No. 3 mine (Frood)	International Nickel Co., Copper Cliff, Ont.	100

III

REPORTS ON THE INVESTIGATIONS CONDUCTED

Test No. 128

TESTS ON THE RECOVERY OF PLATINUM IN BLACK SANDS, BY THE ZACHERT PROCESS

R. K. Carnochan

A shipment of three bags of black sand concentrate, gross weight 465 pounds, was received on November 14, 1919, at the Ore Dressing and Metallurgical Laboratories. This concentrate was from the Bullion mine on the south fork of the Quesnel river, Quesnel district, B.C., and was obtained from the sluice boxes in cleaning up. The concentrate contains gold and platinum and tests were desired to determine if these metals, especially the platinum, could be recovered.

A head sample was taken from the lot and this gave upon assay 0·41 oz. gold per ton and 0·032 oz. platinum per ton. The platinum is very low. To make a higher grade product for test purposes all the concentrate remaining after the removal of the head sample was ground dry in a small ball mill for two hours and then screened on 100 mesh. This gave:

Product	Weight lbs.	Au oz./ton	Pt. oz./ton
+100	107	0·90	0·105
−100	278	0·10	trace

The +100 is high enough in platinum to make experiments on. Two tests were made on this +100, using the Zachert process which consists of amalgamating in the presence of zinc amalgam, copper sulphate and sulphuric acid.

Test No. 1

A lot of 25 pounds 10 ounces of +100 was screened on 40 mesh and the oversize crushed to pass 40 mesh. In doing this some metallics were obtained on the 40 mesh screen. The −40 mesh was then fed to a small amalgamator, and the tailing from the amalgamator was allowed to flow over amalgamation plates. Zinc amalgam had been made in the proportion of 2 parts of zinc to 1 of mercury and then ground to −200 mesh. Twenty-five grams of this amalgam was mixed with the −40 before feeding to the amalgamator, 2½ grams were put into the amalgamator, and 2½ grams were sprinkled on at the top of the plates. Five hundred pounds of water, to which had been added ¼ pound of copper sulphate and ¼ pound of sulphuric acid, were used to wash the −40 through the amalgamator and over the plates.

The tailings from the plates were dried, weighed, and sampled for assay. The mercury was removed from the amalgamator and plates and put through a shammy to obtain amalgam. The amalgam was retorted to obtain the gold and platinum.

Product	Weight lbs.	Au oz./ton	Pt. oz./ton
Amalgam	.....	15·58	0·006
Metallics	.....	0·43	0·065
Tailing	25·19	0·70	0·070
Loss	0·43	.....	.....
Feed	25·62	16·71 0·90	0·141 0·105

The amount of gold in the products is much more than the amount that was in the feed. This is due to some gold remaining on the plates from a previous test on a gold ore. This does not affect the platinum, as no platinum ores had been put over the plates for a long time.

Test No. 2

A lot of 25½ pounds of +100 was treated in the same manner as described under test 1, except that zinc amalgam composed of 9 parts of zinc to 8 parts of mercury was used, and 1¼ pounds of copper sulphate and 1¼ pounds of sulphuric acid were added to the feed water.

Product	Weight lbs.	Au oz./ton	Pt. oz./ton
Amalgam	.....	0·66	0·006
Metallics	.....	0·34	0·056
Tailing	24·00	0·59	0·060
Loss	1·50	.....	.....
Feed	25·50	1·59 0·90	0·122 0·105

SUMMARY

The results given by the two tests are very similar.

The gold recovery in test 2 is 62·9 per cent.

The platinum recovery in test 1 is 50·3 per cent and in test 2 it is 50·8 per cent.

Of the platinum in the feed to the amalgamator there was amalgamated 7·9 per cent in test 1, and 9·1 per cent in test 2.

CONCLUSIONS

1. The Zachert process as used does not give good results on the black sand concentrate.

2. Better results might be obtained by grinding the concentrate finer, say to all through 200 mesh, and in such a way that the amalgamation feed would be scoured and the flakes of gold and platinum brightened up, which would make them easier to amalgamate. This scouring can be secured by using stamps, or by grinding wet in a ball mill before amalgamating.

Further tests under these conditions will be made later.

Test No. 156

RECOVERY OF THE VALUES IN A SILVER-LEAD ORE FROM ARROWHEAD, B.C.

R. K. Carnochan

A shipment consisting of two lots, one of twenty pounds representing what was designated as black sands; and the other of 200 pounds, representing what was designated as yellow sands, was received at the Ore Dressing and Metallurgical laboratories on November 11, 1921, from the Mansfield Mining Company's claims on Trout lake, near Arrowhead, B.C.

A sample cut out from each of these lots gave the following analysis:—

Black sands	Gold,	nil
	Silver,	trace
	Lead,	nil
Yellow sands	Gold,	0·52 oz. per ton
	Silver,	4·92 "
	Lead,	4·40%

The analysis of the black sands, which consisted of graphitic shale, showed practically no values, so that it was not necessary to conduct any further test work on this lot.

The lot designated as yellow sands, which contained lead carbonate and carried values in gold and silver, could be classed as an ore, providing that the cost of mining and metallurgical treatment was below that of the value of the products which could be recovered from it. As this material showed values of about $16 per ton, it had commercial possibilities, and an investigation was undertaken to determine metallurgical methods for the recovery of these values.

Experimental test work was conducted to determine what percentage of recovery of the valuable minerals could be obtained by table concentration; by flotation and table concentration; by table concentration and flotation of the tailings after sulphidizing; by table concentration and cyanidation of the table tailings; by amalgamation; by cyanidation and table concentration.

Test No. 1—Table concentration.
Test No. 2—Flotation and table concentration.
Test No. 3—Table concentration and cyanidation.
Test No. 4—Table concentration and cyanidation.
Test No. 5—Amalgamation.
Test No. 6—Cyanidation and table concentration.

Test No. 1—Tabling

A head sample of ore was ground to pass 20 mesh and screened on 40 and 100 mesh screens. This gave:

	Grams
−20+ 40	483
−40+100	340
−100	405
Total	1,228

Each size was tabled separately on a small laboratory Wilfley table. The slimes from the settling boxes produced in tabling all the sizes were allowed to flow into a large settling tank and were collected as a product. All the products from the table were dried, weighed, and sampled. The following table shows the results obtained:

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
Concentrate—							
− 20+ 40	24	0·56	13·22	33·90	2·03	5·24	15·05
− 40+100	13	3·62	18·44	46·50	7·36	3·97	11·17
−100	12	9·80	55·24	47·20	18·46	10·97	10·47
Tails—							
− 20+ 40	431	0·22	2·48	1·80	14·87	17·68	14·35
− 40+100	289	0·28	3·68	2·70	12·68	17·45	14·42
−100	182	0·60	5·32	4·20	17·06	16·01	14·13
Slimes	88	0·64	6·70	5·80	8·76	9·76	9·43
Slimes loss	189	0·63	6·05	3·14	18·78	18·92	10·98
Heads	1,228	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	27·9
Silver	20·2
Lead	36·7

The results of this test showed that this treatment must be supplemented by other methods to recover more of the values in the ore.

Test No. 2—Flotation and tabling

A head sample of 1,000 grams at −20 mesh was ground with suitable oils in a small ball mill until about 75 per cent would pass 200 mesh. It was then floated in a small Ruth flotation machine, the concentrates being re-run to clean them up. The flotation tailing and middling were mixed and tabled in the manner described in test No. 1. All products were dried, weighed, and sampled.

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
Flotation concentrate	56	4·50	19·96	23·60	48·46	22·73	30·00
Table "	15	3·81	4·58	25·10	10·96	1·40	8·64
" tailing	539	0·10	1·54	1·40	10·38	16·87	17·05
" slimes	362	0·08	6·96	5·40	5·58	51·22	44·31
Slime loss	28	4·57	13·68	.....	24·62	7·78	.....
Heads	1,000	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	59·4
Silver	24·1
Lead	38·6

The results of this test showed that other methods of treatment were necessary to recover more of the valuable mineral in the ore.

Test No. 3—Tabling and cyanidation

A head sample was ground to pass 40 mesh and screened on 100. This gave

	Grams
− 40+100	2,334
−100	2,172

Each size was tabled separately and the slimes produced were put with the −100 tails. The tailings from each size were cyanided separately. The table shows the results obtained. In figuring the table out the assumption was made that the slime loss ran the same in gold and silver as the −100 tailing.

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
+100 concentrate	113	4·32	17·04	43·20	20·83	8·68	24·61
−100 "	88	7·60	21·46	44·00	28·55	8·52	19·52
+100 tails	2,062	0·05	2·35	2·10	4·40	21·86	21·83
−100 "	1,673	0·06	3·74	3·90	4·27	28·22	32·88
Slime loss	570	0·06	3·74	0·40	1·45	9·62	1·16
Cyanided	.....	.....	.....	.....	40·50	23·10	.....
Heads	4,506	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	89·9
Silver	40·3
Lead	44·1

Test No. 4—Tabling and cyanidation

A head sample of 1782 grams −100 mesh was tabled and the tailing and slime produced were mixed and cyanided.

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
−100 concentrate	75	4·72	21·38	33·70	38·19	18·28	32·27
−100 tails	1,344	0·03	1·90	2·28	4·31	29·13	39·03
Slime loss	363	0·03	1·90	6·20	1·19	7·78	28·70
Cyanided	.....	.....	.....	.....	56·31	44·72	.....
Heads	1,782	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	94·5
Silver	63·0
Lead	32·3

Test No. 5—Amalgamation

A head sample of 1,000 grams −40 mesh was amalgamated for 3 hours in a small pebble jar, after which it was panned to recover the mercury. The tailings were dried and sampled. An analysis of these tailings showed them to contain as much gold and silver as the heads, hence, amalgamation on this ore is not possible.

Test No. 6—Cyanidation and tabling

A head sample of 1,000 grams −100 mesh was cyanided and then tabled with the following results:

Tails from cyaniding—	Ozs. per ton
Gold	0·15
Silver	4·10

Recovery by cyaniding—	Per cent
Gold	71·1
Silver	16·7

Tabling—

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
Table concentrate	20	3·28	11·60	40·70	51·56	6·62	20·84
Table tails	468	0·10	2·34	2·60	36·72	31·24	31·15
Table slime	245	0·04	4·48	5·40	7·81	31·32	33·86
Slime loss	122	0·04	8·85	4·53	3·91	30·82	14·15
Feed	855	0·15	4·10	4·57	100·00	100·00	100·00

Recoveries by tabling—

Gold	51·6	per cent of feed to table;			14·9	per cent of heads	
Silver	6·6	"	"	"	5·5	"	"
Lead	20·8	"	"	"	20·8	"	"

Total recoveries—	Per cent
Gold	86·0
Silver	22·2
Lead	20·8

Consumption of cyanide and lime.—In running test No. 6, tests were made of the cyanide solution at different times to determine the amount of chemicals consumed.

The original solution titrated 0·184 per cent KCN, and this fell to 0·168 per cent KCN at the end of the cyaniding.

In the test the ratio of solution to ore was six to one. This gives a consumption of 1·92 pounds of KCN, or its equivalent, per ton of ore.

The original solution titrated 0·086 per cent CaO and at the end of the test, 0·023 per cent CaO. This shows a consumption of 7·56 pounds CaO per ton of ore.

SUMMARY

The recoveries obtained from the various tests are given below:—

Test No.	Initial crushing mesh	Metallurgical treatment	Recoveries of metals, per cent		
			Gold	Silver	Lead
1	20	Table concentration	27·9	20·2	36·7
2	200	Flotation and table concentration	59·4	24·1	38·6
3	40	Table concentration and cyanidation	89·9	40·3	44·1
4	100	Table concentration and cyanidation	94·5	63·0	32·3
5	40	Amalgamation	nil	nil	nil
6	100	Cyanidation and table concentration	86·0	22·2	20·8

CONCLUSIONS

Of the various methods tried for the treatment of this ore, it would seem that the one most applicable is table concentration followed by cyanidation of the table tailings. Test No. 4 shows that by this method a concentrate is obtained assaying, gold, 4·72 ozs.; silver, 21·38 ozs.; lead, 33·7 per cent; and that by cyaniding the table tailings the greater proportion of the remaining gold values are extracted. The total recoveries from this operation are—gold, 94·5 per cent; silver, 63·0 per cent; lead, 32·3 per cent.

In practice, with careful manipulation of machinery, and the working out in detail, the treatment along the lines of procedure of test No. 4, which was conducted on a very small scale, improvement should be made in the grade of the concentrate and the recoveries, as indicated above. On similar ore, as submitted to the department for test purposes, it is safe to assume that 95 per cent of the gold, 65 per cent of the silver, and 45 per cent of the lead values would be recovered.

Test No. 161

THE CONCENTRATION OF THE FLIN FLON DISSEMINATED ORE

C. S. Parsons

In the Summary Report of the Mines Branch for 1921, the results of an investigation on the treatment of the sulphide ore of the Flin Flon ore body was given under test No. 142 of the Ore Dressing and Metallurgical Division. During 1922 an investigation was conducted on the concentration of the disseminated ore of this same ore body, and several small shipments were received from the Mining Corporation of Canada for this purpose.

Dates of shipments.—130 pounds of ore was received February, 1922, and a further shipment of 230 pounds in January 1923.

Analysis of Shipments

——	Cu per cent	Fe per cent	Zn per cent	Au oz.	Ag oz.
Shipment No. 1	2·55	20·05	.....	trace	0·33
Shipment No. 2	2·72	.....	0·49	0·01	0·65

Purpose of investigation.—The purpose of the experimental work was to determine if the ore was amenable to concentration, with the production of a concentrate in as coarse a form as possible, with a high recovery of the copper values. Pyritic smelting of the sulphide ore having been determined as the better practice, the concentrate from the disseminated ore could be mixed with it and bring up the grade of the furnace charge.

Lines of investigation.—On an ore of this class naturally the first line of investigation would be to determine if a copper concentrate could be produced by jigging, removing as much of the copper values as possible in a coarse form. This would be followed by tabling and flotation for the recovery of the remaining copper values more intimately disseminated in the gangue or floating off with the slimes.

If it were found that the copper values were not freed by crushing to a size suitable for jigging, the ore would be further reduced for table concentration, followed by regrinding and flotation for the concentration of finer particles.

If it were found that a table concentrate could not be obtained after crushing to a size suitable for table work, the next line of investigation would be to grind the ore to a fineness at which the copper minerals were free with the production of a copper concentrate by flotation.

Microscopic examination of the disseminated ore.—The microphotographs shown are from a microscopic examination of the ore by Prof. Alfred Wandke, whose report of his investigation brings out the following:—"The sulphide minerals are pyrite, chalcopyrite, and sphalerite. The gangue minerals are calcite, quartz, sericite, and chlorite. Pyrite is the chief sulphide. Chalcopyrite is the chief or only commercial copper mineral. Sphalerite is present, but in much smaller amounts than in the sulphide ore of the same ore body. Aluminous silicates such as chlorite, and sericite, are the chief gangue minerals. It would appear that the disseminated ore would be amenable to a simpler flotation process for the recovery of the copper." This was found to present a more intricate problem, as the results of the experimental test work following will show, due to the presence of deleterious gangue. This could not, however, be foreseen from the microscopic examination.

Experimental Tests

JIGGING

To determine if a coarse concentrate could be obtained from the ore by jigging, a test was made on $\frac{1}{4}''$ material. This test showed conclusively that jigging was not practicable. The concentrate obtained was very low in grade, consisting of all the sulphides. The tailing contained a large amount of unfreed copper mineral.

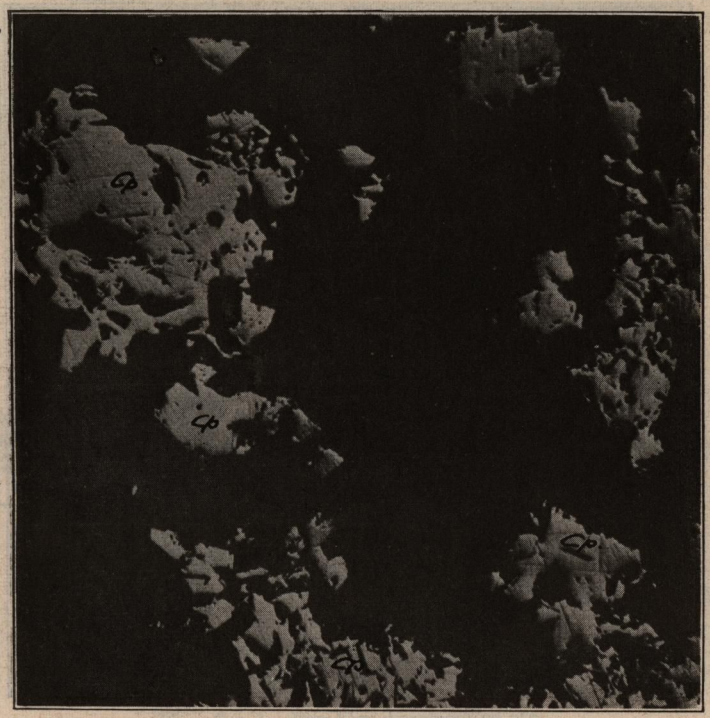
TABLE CONCENTRATION OF THE SIZED MATERIAL

Test No. 1

Object of test. The object of this test was to determine if tabling could be used in conjunction with flotation. It was of course recognized that flotation would have to be used on the fine sizes.

Procedure. 7,000 grams crushed to pass 20 mesh by graded crushing, so as not to produce an excessive proportion of fines. The material was then sized and each size tabled separately. The results are recorded in the accompanying table.

PLATE II.

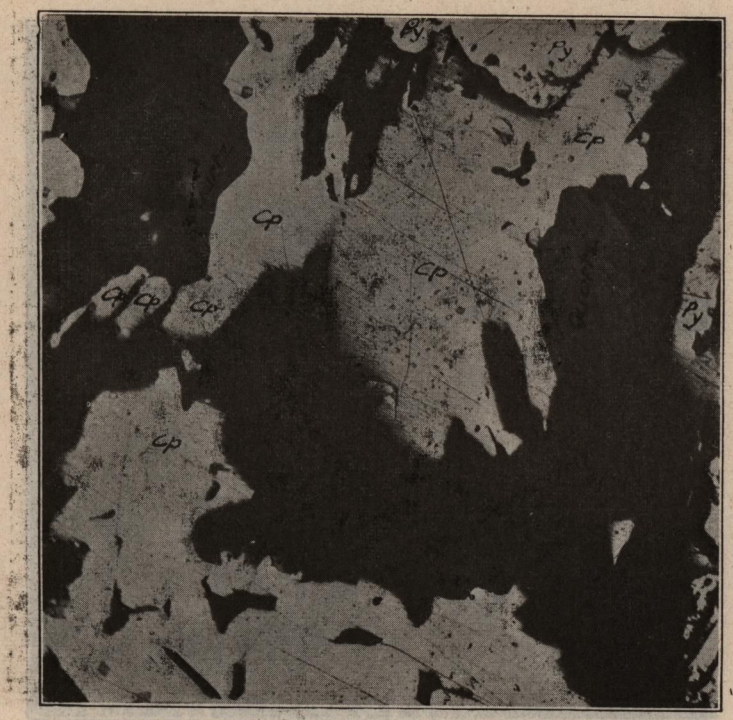


Flin Flon disseminated ore, x 100

Black is chlorite gangue

200 mesh

PLATE III.



Flin Flon disseminated ore, x 100

200 mesh

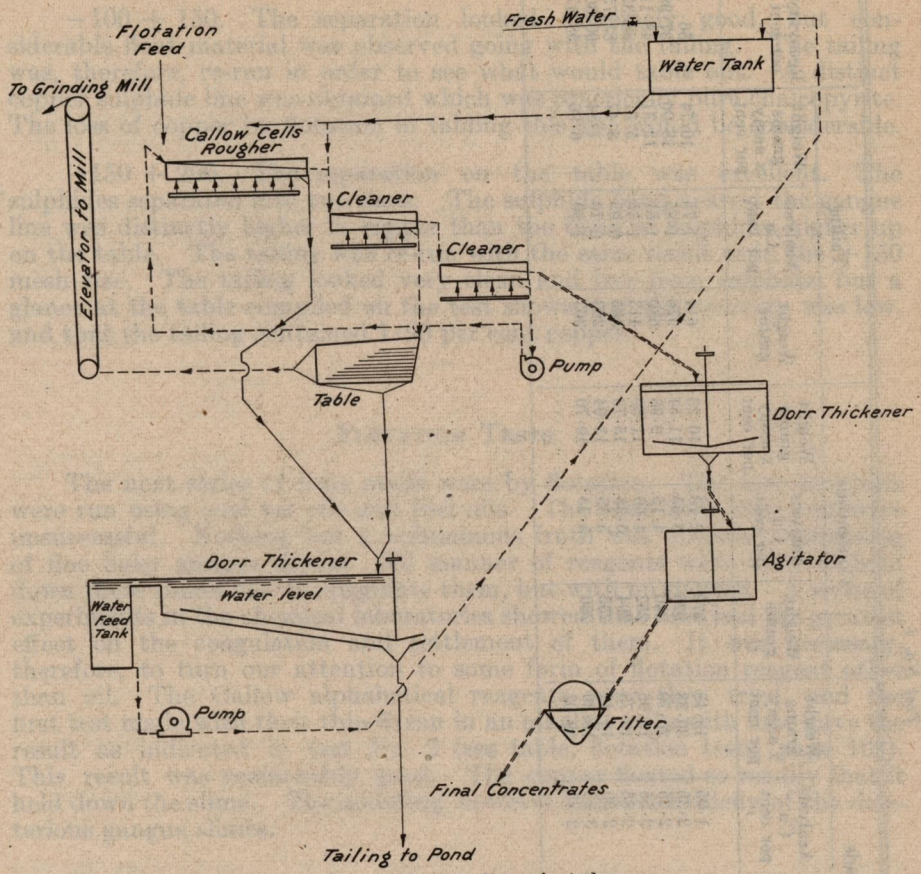


FIG. 4. Flin Flon disseminated ore.

Observations from table:—

−20 + 28. The separation was very poor, and no distinct line of sulphides was obtained.

−28 + 35. The separation was no better.

−35 + 48. The separation of the sulphides showed a marked improvement on this size. The proportion of middling obtained was smaller. The middling was re-run over the table but no separation could be obtained, showing the product was a true middling. The middling was added to the tailing.

−48 + 65. The separation of this size was good. Both the tailing and concentrate looked clean. Very little middling was obtained, and in order to have only two final products the middling was re-run, making a concentrate and tailing, which were added to their respective products obtained from the first run.

−65 + 100. The separation was better than in the previous size.

Test No. 1

Mesh	Heads				Concentrate			Tailing			Recovery per cent
	Weight		Analysis Cu per cent	Weight grams x assay per cent	Weight grams	Analysis Cu per cent	Weight grams x assay per cent	Weight grams	Analysis Cu per cent	Weight grams x assay per cent	
	Grams	Per cent									
+ 28	933	14·3	1·95	18·19	379	2·70	10·23	614	1·75	10·74	56· 2
+ 35	1,293	18·6	2·35	30·38	325	3·55	11·54	976	1·85	18·06	37·98
+ 48	920	13·2	2·25	20·70	240	4·15	9·96	680	1·81	12·31	48·11
+100	659	9·5	2·55	16·96	252	4·60	11·59	413	1·35	5·57	68·33
+150	498	7·1	2·85	18·78	294	5·35	15·73	365	0·95	3·47	83·75
+200	563	8·1	2·90	14·44	235	5·55	13·04	263	0·70	1·84	90·30
−200	1,369	19·7	3·25	18·30	200	5·35	10·70	363	1·90	6·90	58·47
			2·85	39·02							

−100 + 150. The separation looked extremely good, but considerable float material was observed going with the tailing. The tailing was, therefore, re-run in order to see what would table out. A distinct copper sulphide line was obtained which was practically pure chalcopyrite. The loss of copper by flotation in tabling this size would be considerable.

−150 + 200. The separation on the table was excellent. The sulphides separated into two lines. The sulphide band nearest the gangue line was distinctly higher in copper than the band of sulphides higher up on the table. The tailing was re-run with the same result as in the +150 mesh size. The tailing looked very clean and free from sulphide, but a glance at the table compiled on the test shows that the recovery was low, and that the tailing contained 1·90 per cent copper.

Flotation Tests

The next series of tests made were by flotation. The first few tests were run using coal tar oils and fuel oils. The tests with these oils were unsuccessful. Nothing but a voluminous froth was obtained, consisting of fine flaky gangue slimes. All manner of reagents were tried to hold down these slimes and to coagulate them, but with no success. A series of experiments in the chemical laboratories showed that lime had the greatest effect on the coagulation and settlement of them. It was necessary, therefore, to turn our attention to some form of flotation reagent other than oil. The Callow alphabetical reagents were then tried, and the first test made with their thio-fizzan in an alkaline pulp with lime gave the result as indicated in test No. 2 (see table, flotation tests, page 103). This result was remarkably good. The copper floated so readily that it held down the slime. The middling, however, consisted chiefly of the deleterious gangue slimes.

Gravity Test

It was apparent that the talc slime interfered with the flotation. A series of experiments was conducted in an endeavour to reduce the quantity present to a point where it would not interfere with flotation. This deleterious gangue slime seemed to be of a talcy nature or similar to ground up soapstone. It consisted of small flakes which had a greasy or soapy feeling between the fingers. A complete analysis showed these flakes to be of the following chemical composition:—

	Per cent		Per cent
SiO_2	50·0	FeO	11·8
Al_2O_3	3·9	CaO	5·5
MgO	18·8	H_2O	9·5

The gangue, due to the above characteristics, did not settle readily. The following series of gravity tests were run to determine if this property of the gangue could be utilized to separate it from the rest of the ore.

Table Tests

Test No. 3

Object of test:—To determine if the deleterious gangue slimes could be washed out by classification on a Wilfley table and a sand product produced which would be reground and treated by flotation.

Procedure:—Crushing to 20 mesh and tabling without sizing on a small laboratory table. The results are tabulated in the following tables:—

Products	Weight grams	Per cent by weight	Assay per cent Cu	Weight x assay	Per cent of copper values		Remarks
					by feed assays	by product assays	
Concentrate..	885	22·1	4·15	36·73	37·0	36·02	These products retreated by flotation.
Middling.....	1,165	29·1	2·60	30·30	30·5	29·70	
Tailing.....	1,265	31·6	1·70	21·50	21·6	21·07	
Average....	3,315	82·8	2·67	88·53	.....	86·79	
Froth.....	119	3·0	2·50	2·97	3·0	2·90	13·21 per cent of Cu lost in these products.
Slimes.....	317	7·9	2·48	7·86	7·9	7·73	
Loss.....	249	6·3	1·06	2·64	.....	2·58	

Flotation test on above three products, concentrate, middling and tailing, using 1,000 grams of the mixed products

Product	Weight grams	Weight per cent	Assay per cent Cu	Weight x assay	Per cent of Cu values	Remarks
Concentrate.....	201	19·90	11·53	23·17	85·7	Note. This test was only run on 1,000 grams of the total 3,315 grams.
Middling.....	160	15·90	1·00	1·60	5·9	
Tailing.....	647	6·42	0·35	2·26	8·4	

Recapitulation

Product	Weight grams	Weight per cent	Assay per cent Cu	Weight x assay	Per cent of Cu values	Remarks
Concentrate...	662·5	16·5	11·53	76·5	75·0	The recovery is the total recovery of the test.
Tailing.....	3,337·5	83·5	0·76	25·5	25·0	

A loss of 13·21 per cent of the total copper was accounted for in the froth, slimes, and loss in treatment. The froth was the collection of this talc gangue material which floated in the settling tank. The loss is the discrepancy between the amount of material fed, and that recovered in the products.

The recovery in the table sand products, namely, concentrate, middling, and tailing, in the above table was 86·79 per cent of the total copper.

Flotation.—These three sand products were mixed together and a sample weighing 1,000 grams cut out for a flotation test. The ore was ground wet in a ball mill and the test conducted as in test No. 2, using lime and thio-fizzan. A recovery of 85·7 per cent was obtained in the concentrate alone without considering the middling.

Recapitulation.—The recapitulation of the complete test shows that a total recovery of only 75 per cent of the copper was obtained in concentrate assaying 11·5 per cent copper and representing 16·5 per cent by weight of the original ore.

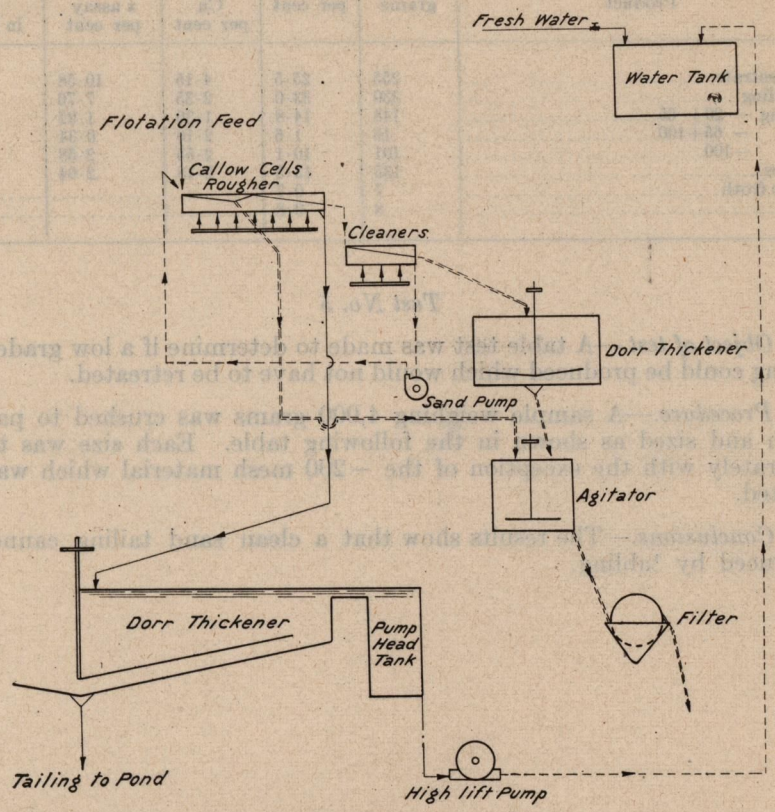


FIG. 5. Flin Flon disseminated ore.

Test No. 4

Object of test.—It was observed in test 3 that considerable fine chalcopyrite floated over with and reported in the sand tailing which otherwise looked clean. The sand tailing from this test was screened to determine if a portion of the sand could be eliminated at this stage as a final tailing.

Procedure.—A 1,000 gram sample was dry crushed in rolls to 20 mesh and tabled without further sizing. A concentrate, middling, and sand tailing was produced together with a slime product which consisted chiefly of the deleterious talcy gangue. The sand tailing was sized on 65 and 100 mesh screens and each product assayed for copper.

It is evident from a glance at the following table that all the sized products contained too high a content of copper to be discarded.

Test No. 4

Product	Weight grams	Weight per cent	Analysis Cu per cent	Weight grams x assay per cent	Per cent of total Cu in heads
Concentrate	255	25·5	4·15	10·58	40·5
Middling	330	33·0	2·35	7·76	29·7
Tailing − 20+ 65	148	14·8	1·30	1·92	7·3
" − 65+100	16	1·6	2·10	0·34	1·3
" −100	101	10·1	2·55	2·58	9·9
Slimes	135	13·5	2·18	2·94	11·3
Table froth	7	0·7	.....	.....	.....
Loss	8	0·8	.....	.....	.....

Test No. 5

Object of test.—A table test was made to determine if a low grade sand tailing could be produced which would not have to be retreated.

Procedure.—A sample weighing 4,000 grams was crushed to pass 20 mesh and sized as shown in the following table. Each size was tabled separately with the exception of the −200 mesh material which was not treated.

Conclusions.—The results show that a clean sand tailing cannot be produced by tabling.

Test No. 5—Run to make clean tailing

Mesh	Heads			Concentrate				Middling				Tailing			
	Weight		Calcul. analysis Cu	Weight grams	Analysis Cu	Weight x assay	Recovery	Weight grams	Analysis Cu	Weight x assay	Per cent of Cu values	Weight grams	Analysis Cu	Weight x assay	Per cent of Cu values
	Grams	Per cent	per cent		per cent	per cent	per cent		per cent	per cent			per cent	per cent	
+ 28	731	14·91	2·51	193	3·78	7·3	39·8	374	2·43	9·1	49·5	164	1·20	2·0	10·7
+ 35	1,062	21·64	2·49	379	3·75	14·2	53·7	503	2·08	10·5	39·5	180	1·00	1·8	6·8
+ 48	653	13·32	2·64	147	4·15	6·1	35·3	331	2·80	9·3	53·7	175	1·08	1·9	11·0
+ 65	441	8·99	2·84	100	4·58	4·6	36·6	194	3·30	6·4	51·1	147	1·05	1·5	12·3
+100	415	8·05	3·12	95	4·85	4·6	35·5	172	3·60	6·2	47·7	148	1·47	2·2	16·8
+200	643	13·11	3·38	164	5·10	8·4	38·4	191	3·95	7·5	34·7	288	2·03	5·8	26·9
−200	906	18·48	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
Table slimes	52	1·06	2·60	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Test No. 7

Object of test.—This test was to determine if the objectionable talcy material could be eliminated in a tailing without excessive loss of copper. It was thought that perhaps wet crushing would free this talcy gangue material better than dry crushing which had been used in the preceding tests.

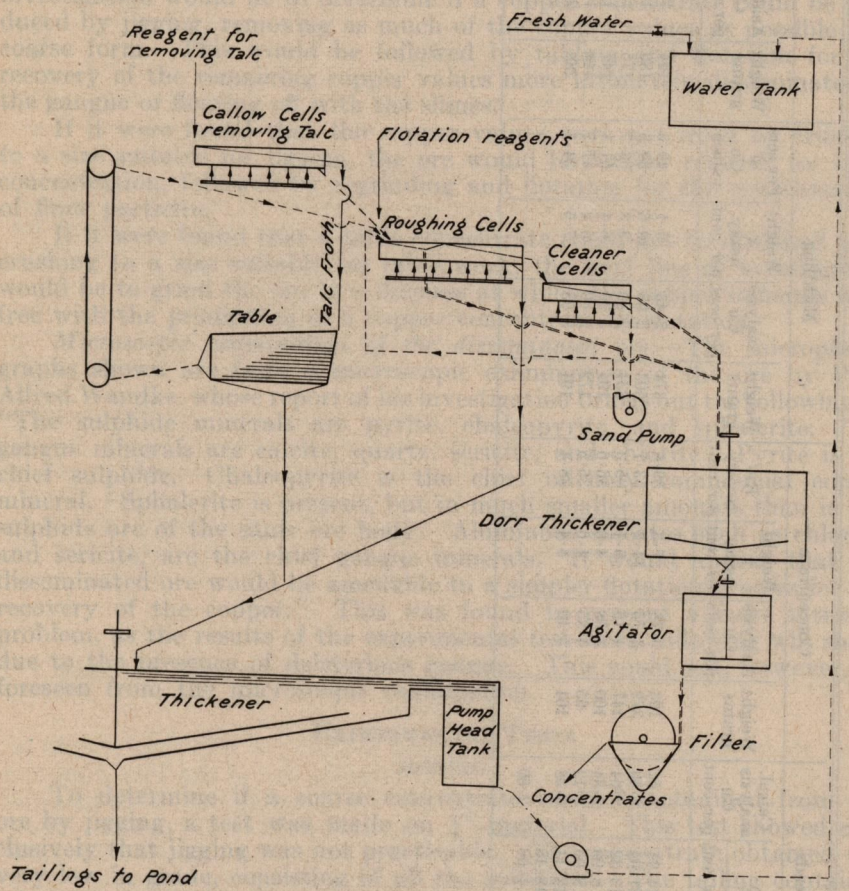


FIG. 6. Flin Flon disseminated ore.

Procedure.—A 1,000 gram sample of ore was prepared by dry crushing to pass 20 mesh and then placed in a wet ball mill for a five-minute period for further crushing. This crushed product was then tabled. The results are recorded in the accompanying table. The slimes contained 12·6 per cent of the total copper, and from observation it was noted that the sand tailing still contained considerable of the talcy material.

Product	Weight grams	Weight per cent	Analysis Cu per cent	Weight x assay per cent	Per cent of Cu values
Concentrate	268	26·8	5·10	13·7	51·1
Middling	145	14·5	4·10	5·9	22·2
Tailing	400	40·0	0·94	3·8	14·1
Slime	187	18·7	1·80	3·4	12·6

Test No. 8

Object of test.—Same as in test No. 7.

Procedure.—The ore was not crushed as fine as in the preceding test in the hope that less copper would be lost in the slimes. The loss of copper in the slimes containing the deleterious talcy material was approximately 4 per cent less than in test No. 7.

Product	Weight grams	Weight per cent	Analysis Cu per cent	Weight x assay per cent	Per cent of Cu values
Concentrate	393	39·3	5·00	19·6	73·8
Tailing	461	46·1	1·00	4·6	17·3
Slime	131	13·1	1·78	2·3	8·8
Loss	15	1·5	.....	.....	.....

Elimination of Objectionable Talcy Material by Hydraulic Classification

A series of tests, Nos. 10, 11, 12, 13, 18, 23, 24 and 25, were made using hydraulic classification after comparative coarse crushing to eliminate the slimes interfering with flotation.

Summary of results.—The products showing the results of these tests are tabulated in tables following. The variation in the amounts of slime removed is due to the difficulty in controlling the quantity of slime overflow from the Richards hydraulic classifier.

For instance, in test 10, too large a quantity of slime was taken off, approx. 44·1 per cent. The amount separated as slimes in the other tests varied between 12 per cent and 18 per cent of the total feed. The variations in the total recovery of copper in concentrate is due to erratic results from flotation. The reason for this variation has not been determined. The flotation results are apparently more erratic when the slimes are removed than when they are present in the flotation pulp.

Test No. 10

Object of test.—To determine if stage crushing with classification after each reduction would produce a sand product free from the objectionable slimes.

Procedure.—1,000 grams of ore $-\frac{3}{8}''$ crushed for 5 minutes in small ball mill and resulting product classified by panning. This was repeated twice. The sands were reground to 100 mesh for flotation and the separated slimes tabled. The reagents used for flotation were thio-fizzan and lime.

Test No. 11

Object of test.—The elimination of the deleterious slime by crushing and classification in order to produce a product suitable for flotation.

Procedure.—1,000 grams of ore $-\frac{1}{4}''$ crushed in wet ball mill for 5 minutes and the sands and slime of resulting product separated in Richards classifier. The sands reground to 100 mesh for flotation. Reagents used for flotation were thio-fizzan and lime.

Test No. 12

A duplicate of test No. 11, with the exception that the separated slimes were tabled making concentrate and tailing.

Test No. 13

A duplicate of tests 10 and 11, with the exception that the slimes from classification were treated by flotation in high pulp dilution. Note: The reagents used for tests 12 and 13 were thio-fizzan and lime.

Tests Nos. 18, 23, 24 and 25

These tests were run as duplicate check tests.

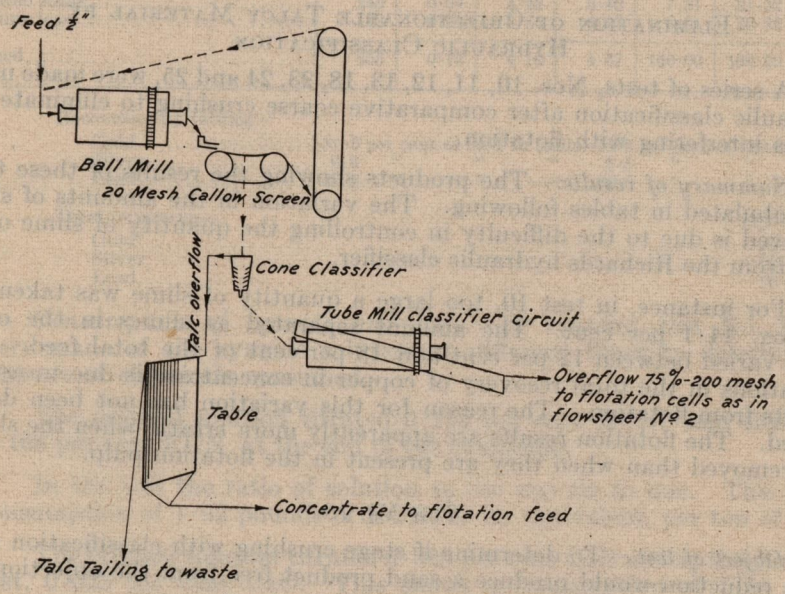


FIG. 7. Flin Flon disseminated ore.

Test No.	Product	Weight		Copper		Per cent of values	Remarks
		Grams	Per cent	Per cent	Weight x assay		
10	Table slimes....	208	20·8	1·68	3·49	13·7	
	" tailg.....	158	15·8	1·07	1·69	6·6	
	" conc.....	75	7·5	4·80	3·60	14·1	
	Flot. tailg.....	259	25·9	0·52	1·34	5·3	
	" midd.....	71	7·1	0·88	0·62	2·4	
	" conc.....	212	21·2	6·95	14·75	57·8	
	Loss.....	12	.....	.....	.....	.....	
11	Flot. conc.....	125	12·5	13·78	17·22	67·3	
	" midd.....	98	9·8	2·00	1·96	7·7	
	" tailg.....	595	59·5	0·53	3·15	12·4	
	Slime tailg.....	135	13·5	1·05	1·43	5·6	
	Loss.....	47	4·7	.....	.....	6·9	(Estimated.)
12	Flot. conc.....	163	16·3	11·16	18·19	76·8	Recovery in table concentrate and flotation concentrate, 82·4 per cent
	" midd.....	147	14·7	1·35	1·98	8·4	
	" tailg.....	517	51·7	0·30	1·55	6·5	
	Table conc.....	25	2·5	5·30	1·33	5·6	
	" tailg.....	84	8·4	0·75	0·63	2·7	
	Loss.....	64	6·4	.....	.....	.....	
13	Flot. conc. No. 2	214	21·4	9·92	21·33	81·7	Grade of concentrate from two concentrates, 8·71 per cent Cu.
	" " No. 1	45	4·5	2·74	1·23	4·7	
	Total conc.....	259	25·9	8·71	22·56	86·4	
	Flot. Midd. No. 2	174	17·4	1·12	1·94	7·4	
	" tailg. No. 2	492	49·2	0·27	1·33	5·1	
	" tailg No. 1	32	3·2	0·88	0·28	1·1	
	Loss.....	43	4·3	.....	.....	.....	
23c	Flot. conc. No. 1	103	10·3	12·07	12·44	49·0	Ore crushed dry $\frac{3}{4}$"; wet in ball mill to 6 mesh; after separating in classifier sands reground to 65 mesh and floated. Slimes floated in very dilute pulp. KK oil used in this test.
	" " No. 2	85	8·5	4·57	3·89	15·3	
	" midd.....	134	13·4	2·96	3·97	15·6	
	" tailg. No. 1	474	47·4	0·75	3·55	14·0	
	" " No. 2	168	16·8	0·92	1·55	6·1	
	Loss.....	36	3·6	.....	.....	.....	
24c	Flot. conc. No. 1	141	14·1	13·55	19·25	74·0	81 per cent.
	" " No. 2	37	3·7	4·92	1·82	7·0	
	" midd.....	155	15·5	1·57	2·43	9·4	As in test 23c.
	" tailg. No. 1	542	54·2	0·32	1·73	6·7	
	" " No. 2	101	10·1	0·75	0·76	2·9	
	Loss.....	24	2·4	.....	.....	.....	
25c	Flot. conc. No. 1	106	10·6	15·40	16·39	66·9	
	" midd. No. 1	117	11·7	2·12	2·38	9·7	
	" tailg. No. 1	563	56·3	0·40	2·25	8·8	As in test 23c.
	" conc. No. 2 (Re-run classifier fines).....	38	3·8	5·66	2·44	10·0	
	Flot. tailg. No. 2	126	12·6	0·87	1·10	.....	
	Loss.....	50	5·0	.....	.....	.....	
18	Flot. conc. No. 1	101	10·1	17·50	17·33	65·0	
	" midd. No. 1	105	10·5	2·47	2·35	8·8	
	" tailg. No. 1	654	65·4	0·77	5·04	18·9	As in test 23c.
	" conc. No. 2 (Re-run classifier fines).....	19	1·9	6·86	1·30	4·9	
	Flot. tailg. No. 2	73	7·3	0·90	0·65	2·4	
	Loss.....	48	4·8	.....	.....	.....	

Continuation of Flotation Tests

It had already been found that oils could not be used. With oils, such large quantities of the talc froth are formed that it prevents the copper from floating.

Two methods of procedure were next tried. The first, straight flotation of the ore using alphabetical reagents, and the second, removing the talc by a preliminary flotation with a frothing reagent.

The first method is the simpler process, but the talc interferes seriously in the production of a high grade concentrate. The second method gives satisfactory results in the laboratory, but the possibility of certain difficulties in the application of this method to practice can be foreseen. One of the chief difficulties would be that of preventing this part of the flotation circuit from becoming contaminated by collecting oils, such as lubricating oils, etc., which would cause large losses of copper in the froth product. The froth produced is also sticky and difficult to handle.

Chemical reagents.—It has been previously mentioned that lime was found to be the most satisfactory reagent to hold down the talc slimes. A large number of salts were tested in the laboratory and their effect on the coagulation and settlement of this talc material compared. Lime was found to coagulate and settle these slimes more rapidly than any other reagent tried.

What actually happens when flotation of the copper is attempted in the presence of the talcy substances, is that the material being in the form of flat thin flakes, presents a large surface area, and is present in relatively large quantities compared to the copper, and so completely armours the bubbles and excludes the chalcopyrite from their interfacial boundaries.

Removal of talc by preliminary flotation.—Referring to the second scheme for removing the talc by the use of a frothing reagent which had no collecting action on the copper, we found that Fumol No. 6, a product of the Canadian Electro Products Co., Shawinigan Falls, Que., gave the most satisfactory results. It is possible that other similar reagents might give improved results. This phase of the problem was discussed with the chemists of the Canadian Electro Products Co., who undertook to prepare some separate samples. Nine samples of Aldol and mixtures of Aldol and Fumol products were prepared. We suggested the mixing of sulphur in some of their products and three such samples were prepared. Duplicates of all these samples with full particulars as to how they were made were kept by the company. Sample No. 22a is Aldol saturated with SO_2. No. 24a is Fumol saturated with SO_2, and No. 27 is Aldol saturated with H_2S. None of the nine samples gave any better results than the standard Fumol No. 6. For reference see tests Nos. 44 to 54.

Size of crushing.—The removal of this deleterious gangue by flotation introduced the question that perhaps the size of crushing had an important effect on the results. A series of tests were run, the ore being crushed to different sizes. When the ore was crushed through 100 mesh, and when approximately 75 per cent would pass 200 mesh, the best condition for flotation was obtained both as regards the removal of the deleterious gangue and the subsequent recovery of the copper.

General Procedure of Tests for Flotation.—The ore was crushed by dry crushing in rolls to 20 mesh. A 1,000 gram lot was then cut out for each test and placed in a small ball mill. If lime was used it was added in the ball mill. Experience has shown that this is the only satisfactory place to add the lime, as it is important for the lime to be in contact with the ore for considerable time previous to flotation. A pulp, one of solids to one of water, was made up and the charge ground for the required time to produce the desired size of product. The flotation tests unless otherwise mentioned were conducted in the Ruth laboratory machine. We find this the simplest machine to operate for test work. If the talc was to be removed by preliminary flotation, the frothing reagent was added in the cell, and after the bulk of the talc had been removed, the reagent for floating the copper was added.

Tests by General Engineering Co.

A small sample of ore was sent to the General Engineering Co., who tried two methods of treatment to check our results. An abstract from their report reads as follows:—

"Two methods of treatment were tried. The first by straight flotation with a frothing reagent, tabling the talc product so obtained, to remove any of the copper which floated with it, returning the table concentrate to the flotation circuit, and floating the pulp from which the bulk of the talc has now been removed for copper.

"The first method is much the simpler flow sheet, but the concentrate produced is lower grade. The second method gives higher grade concentrate and should show about the same recovery, although I was not able to do this. I can offer no explanation as to why as much copper did not float after removing the talc as before removing it.

"On the whole my tests by straight flotation show a slightly better tailing and recovery, and a lower grade concentrate than theirs (Mines Branch), but their recovery when removing the talc first is better than mine. About 8 pounds per ton lime seems to give the proper alkalinity for either method. In the straight flotation tests this is ground with the ore. When the talc is removed first, the lime is added to the pulp after the talc flotation. Grinding 1%+100 mesh and 80%−200 mesh gave as good results as 95% −200 mesh. T.T. mixture and thio-fizzan gave about the same results. Grinding thiocarbanilid with the ore gave no results."

"The accompanying tables show the results of the best tests by each method. These tests were made in the Callow laboratory testing cells. In the operation of the cells, the middling from the cleaner cell is returned to the rougher cell and retreated."

RESULTS OF TESTS BY GENERAL ENGINEERING CO.

Ground with 8 lbs. lime, floated with ⅛ lb. of thio-fizzan.

Product	Per cent weight	Per cent copper	Per cent recovery
Concentrate	21·9	10·48	87·0
Tailing	78·1	0·44	13·0

Floated with $\frac{1}{10}$ lb. pine oil to remove talc, tabled talc product, added this table concentrate to the pulp remaining from talc flotation, floated with 8 lbs. lime, $\frac{4}{10}$ lb. TT mixture and $\frac{1}{12}$ lb. Aldol.

Product	Per cent weight	Per cent copper	Per cent recovery
Flotation concentrate	14·5	14·74	80·1
" tailing	66·5	0·67	16·7
Talc tailing	19·0	0·45	3·2
Combining the two tailings			
Flotation concentrate	14·5	14·75	80·1
Tailing	85·5	0·62	19·8

Same flow sheet and reagents as above except $\frac{1}{4}$ lb. thio-fizzan replaces TT and Aldol in floating copper. High grade concentrate obtained.

Product	Per cent weight	Per cent copper	Per cent recovery
Flotation concentrate	10·9	18·68	76·2
" tailing	65·5	0·81	19·8
Talc tailing	23·6	0·44	4·0
Combining the two tailings			
Flotation concentrate	10·9	18·68	76·2
Tabling	89·1	0·71	23·8

Tests made in the Callow laboratory cells.—After receiving the report from the General Engineering Co., tests Nos. 38 and 37b were run in our Callow testing unit to check their results. We found that we were still able to obtain a better recovery than they when removing the talc first, but that our recovery by straight flotation is lower than theirs.

Comparison and discussion of results in Ruth and Callow machines.—The recovery in the Ruth machine, under favourable conditions is higher than in the Callow. A 0·25 per cent copper tailing was consistently obtained in the Ruth against a 0·40 per cent or higher tailing in the Callow when using straight flotation. When the talc was removed by preliminary flotation we obtained approximately the same results in both machines.

In the Callow unit the middling from recleaning the rougher concentrate is returned continuously to the rougher cells, thus increasing the amount of talc present in that cell because a large proportion of the middling returned consists of the talc material. The operation of the Ruth is entirely different. A rougher concentrate is made and a final tailing in the first operation. The cell is then cleaned out, after which the rougher concentrate is placed in it and fresh water added to fill the cell. The rougher concentrate is then recleaned by a separate operation producing a final concentrate and a middling product. This middling product is kept separate and not added back to rougher cell, so that the circuit is not further contaminated by the presence of additional quantities of talc.

Accumulative middling test in Ruth machine.—An accumulative middling test was made to determine the effect of returning the middling to the rougher cell. This is test No. 36. The procedure followed was to start off with a lot of 1,000 grams of ore reduced to the required size. This lot was floated while a second lot was being prepared. The middling from the recleaning of the concentrate from the first lot was then added to the second lot and the flotation proceeded with as before. This was repeated until five lots of ore had been run. If the assays from the products resulting

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from this test are studied it will be evident that returning the middling to be mixed with the feed to the rougher cells is detrimental. The middling builds up in the circuit and loads the circuit up with froth, finally reducing the grade of the concentrate, and increasing the loss in the tailing.

Test No. 36

Product	Weight grams	Copper	
		Per cent	Per cent x grams
Concentrate No. 1	108	20·42	22·06
Tailing No. 1	820	0·45	3·69
			25·75
Concentrate No. 2	117	16·40	19·18
Tailing No. 2	822	0·85	6·99
			26·17
Concentrate No. 3	107	19·05	20·39
Tailing No. 3	833	0·60	5·00
			25·39
Concentrate No. 4	105	18·15	19·06
Tailing No. 4	1,011	1·15	11·63
			30·69
Concentrate No. 5	81	12·15	9·84
Tailing No. 5	910	1·75	15·93
			25·77

Explanation of lower tailing in Ruth machine.—The assumption is correct that lower tailings are obtained in the Ruth than in the Callow machine, for the reason that in the Callow, the middling is returned continuously to the rougher cell.

Separate treatment of middling.—The conclusion has been reached, based on the results from the above experiments that the middling from the cleaner cell should be either treated in a separate circuit or tabled to remove the talc before being returned to the flotation circuit. It may possibly be found more advantageous to regrind the table concentrate before returning to the flotation machines.

SUMMARY OF CONCLUSIONS BASED ON RESULTS OF WORK TO DATE

The work up to the present has shown that there are four possible flow sheets for the treatment of this ore, as shown by Figures 4, 5, 6 and 7.

1. Straight flotation, but middling tabled to remove the talc before being returned to the circuit as suggested under the above heading "Separate treatment of the middling."

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2. Straight flotation with middling returned to circuit.

3. Removal of the talc by preliminary flotation with a reagent which will float the talc without having a collecting action on the copper, followed by flotation of the copper from the pulp after the removal of the bulk of the talc.

4. Removal of the bulk of the talc by hydraulic classification at different points in the grinding circuit and flotation of the modified pulp as in (1). (See Figs. 4, 5, 6 and 7.)

The first proposed flow sheet appears at present to be the most promising one. No work so far has been done on this. The second flow sheet the writer believes comes next in order. The results from the straight flotation tests have been erratic. The reason for this has so far not been determined. However, it is safe to assume that a 10 per cent copper concentrate with an 80-85 per cent recovery can be obtained. The third method is probably the next best, but it is hardly possible by laboratory experiments to determine how successfully the flow sheet could be used in practice. Our laboratory experiments show that a very high grade concentrate can be obtained after the talc is removed, containing 18-25 per cent copper with an 80 per cent or better recovery. What has been said about the third method applies also to the fourth.

Proposed Lines on Which Research Will be Continued

1. Straight flotation but with middling tabled to remove the talc before returning to the circuit.—Considerable detailed work will be done on this method of treatment as at present it appears to be the most promising flow sheet for the concentration of this ore. Careful attention will be given the regrinding of the middling before it is returned to the flotation circuit.

2. Straight flotation.—A number of experiments will be run using larger quantities of ore under conditions approaching as nearly as possible actual milling operations. Because the results of the straight flotation tests have been very erratic, a series of experiments will be run under careful temperature control. Both the temperature in the ball mill while grinding, and in the flotation machine will be considered. Other variables will be kept under close observation in an endeavour to determine the correct conditions governing the production of uniform results.

3. Removal of the talc by preliminary flotation.—Very little more work will be done on this method. It has been clearly demonstrated that this can be done successfully in the laboratory. A very high grade concentrate can be obtained, apparently higher than with any of the other proposed flow sheets. We find the recovery to be about the same as with the other methods, but the General Engineering Co. report that it is lower. Some further work will be done to clear up this point.

4. Removal of the bulk of the talc by hydraulic classification.—We consider it very difficult to obtain results in the laboratory applicable to milling operations, and, therefore, do not propose doing much further work on this flow sheet.

Flotation Tests

Test No.	Product	Weight		Copper		Per cent of Cu values	Remarks
		Grams	Per cent	Per cent	Per cent x weight		
2	Concentrate.....	127	12·7	16·90	21·46	85·6	0·10 lb./ton thio-fizzan, 4 lb. lime; 65 mesh, ground wet.
	Middling.....	165	16·5	1·75	1·89	7·4	
	Tailing.....	708	70·8	0·25	1·77	7·0	
14	Concentrate.....	262	26·2	7·97	20·88	79·3	1 lb. KK oil and 2 lb. lime; 100 mesh, froth from gangue.
	Middling.....	230	23·0	1·23	2·83	10·7	
	Tailing.....	507	50·7	0·52	2·64	10·0	
16	Concentrate.....	112	11·1	17·82	19·78	77·7	As in test 2.
	Middling.....	135	13·3	1·13	1·52	6·0	
	Tailing.....	766	75·6	0·55	4·16	16·3	
19a	Concentrate.....	101	10·0	20·62	20·83	77·0	Thio-fizzan and 4 lb. lime.
	Middling.....	108	10·6	1·85	1·98	7·3	
	Tailing.....	807	79·4	0·53	4·28	15·7	
20a	Concentrate.....	108	10·3	19·17	20·70	76·3	Thio-fizzan and 8 lb. lime.
	Middling.....	227	21·6	1·10	2·29	8·5	
	Tailing.....	717	68·1	0·57	4·09	15·1	
26	Concentrate.....	156	15·5	11·60	18·10	64·6	−48 mesh; 44 per cent −200 mesh, thio-fizzan and 5 lb. lime.
	Middling.....	177	17·6	2·27	5·08	18·1	
	Tailing.....	673	66·9	0·72	4·85	17·3	
27	Concentrate.....	145	14·5	13·62	19·75	76·8	−65 mesh; 53 per cent −200 mesh; thio-fizzan and 5 lb. lime.
	Middling.....	146	14·6	1·50	2·19	7·5	
	Tailing.....	712	70·9	0·57	4·06	15·7	
28	Concentrate.....	121	11·9	18·67	22·60	86·4	−100 mesh; 70 per cent −200 mesh; thio-fizzan and 5 lb. lime.
	Middling.....	136	13·4	0·40	0·54	2·0	
	Tailing.....	761	74·7	0·40	3·04	11·6	
29	Concentrate.....	141	14·0	14·92	21·04	80·1	−100 mesh; 82 per cent −200 mesh; thio-fizzan and 5 lb. lime.
	Middling.....	151	15·0	1·57	2·37	9·0	
	Tailing.....	715	71·0	0·40	2·86	10·9	
31	Concentrate.....	109	10·8	20·22	22·04	83·0	−100 mesh; 82 per cent −200 mesh; TT mixture and 8 lb. lime.
	Middling.....	90	9·0	2·15	1·94	7·3	
	Tailing.....	808	80·2	0·32	2·59	9·7	
33	Concentrate.....	113	11·2	18·02	20·36	77·3	YZ mixture and lime.
	Middling.....	207	20·5	2·00	4·14	15·7	
	Tailing.....	692	68·3	0·25	1·73	6·6	
35	Concentrate.....	125	12·5	12·77	15·84	60·7	
	Middling.....	195	19·5	2·20	4·29	16·5	
	Tailing.....	684	68·0	0·87	5·95	22·8	
37	Concentrate.....	149	14·9	12·80	19·07	74·3	
	Middling.....	180	18·0	1·25	2·25	8·8	
	Tailing.....	671	67·1	0·65	4·36	16·9	
38	Concentrate.....	160	15·4	13·08	20·93	75·3	TT mixture and 6 lb. lime per ton. Callow cells.
	Clean up cells...	40	3·9	4·55	1·82	6·5	
	Tailing.....	840	80·7	0·60	5·04	18·2	
15	Concentrate.....	116	11·4	18·92	21·95	81·1	2 lb./ton lime; added Aldol to cell, took of froth, added thio-fizzan.
	Middling.....	115	11·4	1·42	1·63	6·0	
	Tailing.....	587	58·0	0·33	1·94	7·2	
	Froth.....	194	19·2	0·80	1·55	5·7	

Flotation Tests—Continued

Test No.	Product	Weight		Copper		Per cent of Cu values	Remarks
		Grams	Per cent	Per cent	Per cent x weight		
21b	Concentrate.....	144	14·0	15·87	22·88	83·3	
	Middling.....	116	11·3	1·10	1·28	4·7	Same as in test 15.
	Tailing.....	565	54·7	0·22	1·24	4·5	
	Froth.....	207	20·0	0·96	1·99	7·3	
22b	Concentrate.....	134	12·6	16·27	21·80	81·0	
	Middling.....	125	11·8	1·25	1·56	5·8	Same as in test 15.
	Tailing.....	655	61·7	0·34	2·23	8·3	
	Froth.....	147	13·9	0·90	1·32	4·9	
37b	Concentrate.....	148	14·7	15·78	23·40	83·3	Same as in test 15. Run in small Callow.
	Clean up cells...	62	6·2	1·25	0·78	2·8	
	Tailing.....	576	57·2	0·35	2·02	7·2	
	Froth.....	221	21·9	0·85	1·88	6·7	
30	Concentrate.....	98	9·9	22·40	21·95	82·5	As in above tests. Z- cake present during frothing.
	Middling.....	117	11·8	1·10	1·29	4·8	
	Tailing.....	596	60·1	0·25	1·49	5·6	
	Froth.....	180	18·2	1·05	1·89	7·1	
17	Flot.concentrate	97	9·7	21·02	20·42	82·7	As above; froth was tabled.
	" middling..	75	7·5	1·47	1·10	4·5	
	" tailing.....	589	58·9	0·22	1·30	5·3	
	Re-run flotation froth—						
	Table concent'te	16	1·6	2·70	0·31	1·5	
	" tailing...	200	20·0	0·76	1·52	6·2	
	Loss.....	23	2·3	.....	.....	.....	
40	Concentrate.....	177	17·7	10·00	17·70	64·6	Lime, 4 lb./ton; 0·1 cc. TT; 1 drop thio-fizzan.
	Middling.....	193	19·3	3·55	6·85	25·0	
	Tailing.....	630	63·0	0·45	2·83	10·4	
40a	Concentrate.....	75	15·1	12·75	9·50	71·9	As test 40, but 500g; 4 lb./ton lime; 0·05 c.c. TT; ½ drop thio-fizzan.
	Middling.....	51	10·3	3·30	1·58	12·6	
	Tailing.....	371	74·6	0·55	2·06	15·5	
41	Concentrate.....	180	21·6	10·15	22·80	79·4	4 lb./ton lime; ½ drop c.t.c., 2 drops thio-fizzan—ground.
	Middling.....	83	10·0	2·15	2·15	7·5	
	Tailing.....	570	68·4	0·55	3·76	13·1	
42	Concentrate.....	79	15·9	12·05	19·16	73·6	4 lb./ton lime; 2 drops thio-fizzan; ½ drop c.t.c. and c.t. mixture.
	Middling.....	126	25·4	2·00	5·08	19·6	
	Tailing.....	291	58·7	0·30	1·76	6·8	
44	Froth.....	144	14·4	0·90	1·29	4·9	Gangue frothed with Fumol No. 21. Added 2 grams soda ash and 0·15 c.c. TT.
	Concentrate.....	106	10·6	18·55	19·67	74·5	
	Middling.....	126	12·6	2·85	3·59	13·5	
	Tailing.....	623	62·4	0·30	1·87	7·1	
45	Froth.....	245	24·3	1·12	2·70	10·3	Lime, 6 lb./ton. Gangue frothed with Fumol No. 6, 3 drops. Floated with TT 0·15 c.c. .....
	Concentrate.....	101	10·0	20·27	20·47	78·4	
	Middling.....	138	13·7	1·75	2·43	9·3	
	Tailing.....	523	52·0	0·10	0·52	2·0	
46	Froth.....	96	9·6	1·45	1·39	5·3	As No. 44, but Fumol No. 20.
	Concentrate.....	121	12·1	16·95	20·51	78·6	
	Middling.....	143	14·2	1·68	2·40	0·2	
	Tailing.....	644	64·1	0·28	1·80	6·9	

Flotation Tests—Concluded

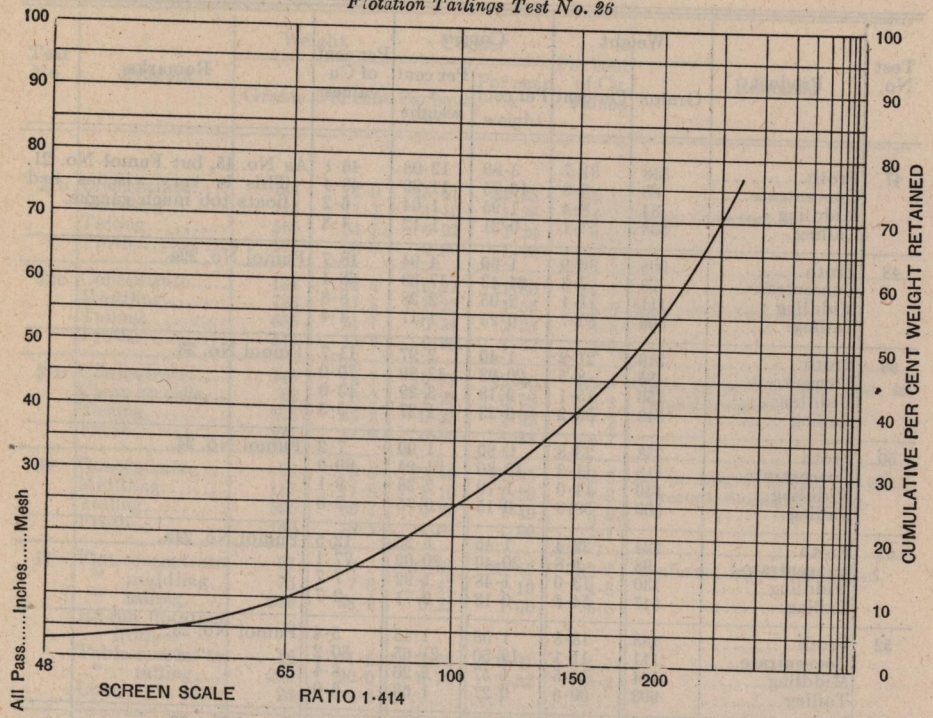
Test No.	Product	Weight		Copper		Per cent of Cu values	Remarks
		Grams	Per cent	Per cent	Per cent x weight		
47	Froth	318	31·7	3·80	12·08	46·1	As No. 45, but Fumol No. 21. This is very viscous and floats too much gangue.
	Concentrate	68	6·8	16·75	11·39	43·4	
	Middling	84	8·4	1·95	1·64	6·2	
	Tailing	533	53·1	0·21	1·12	4·3	
48	Froth	308	30·9	1·60	4·94	18·7	Fumol No. 22a.
	Concentrate	73	7·3	24·43	17·83	68·1	
	Middling	111	11·1	2·05	2·28	8·8	
	Tailing	506	50·7	0·23	1·17	4·4	
49	Froth	210	21·2	1·40	2·97	11·7	Fumol No. 23.
	Concentrate	85	8·5	20·93	17·80	70·0	
	Middling	150	15·1	2·18	3·29	13·0	
	Tailing	548	55·2	0·23	1·27	5·3	
50	Froth	238	23·8	0·80	1·90	7·2	Fumol No. 24.
	Concentrate	113	11·3	18·80	21·24	80·9	
	Middling	140	14·0	1·70	2·38	9·1	
	Tailing	500	50·0	0·15	0·75	2·8	
51	Froth	224	22·4	1·45	3·25	12·5	Fumol No. 24a.
	Concentrate	98	9·8	20·40	20·02	77·1	
	Middling	130	13·0	1·48	1·92	7·7	
	Tailing	547	54·8	0·13	0·71	2·7	
52	Froth	138	13·8	1·05	1·45	5·4	Fumol No. 25.
	Concentrate	111	11·1	19·50	21·65	80·2	
	Middling	144	14·5	1·57	2·26	8·4	
	Tailing	603	60·6	0·27	1·63	6·0	
53	Froth	206	20·3	2·12	4·37	16·0	Fumol No. 26.
	Concentrate	99	9·8	20·05	19·85	72·6	
	Middling	154	15·1	1·32	2·03	7·4	
	Tailing	553	49·8	0·20	1·11	4·0	
54	Froth	100	9·9	6·09	6·09	21·8	Fumol No. 27.
	Concentrate	144	14·2	12·40	17·86	64·4	
	Middling	158	15·6	1·52	2·40	8·7	
	Tailing	609	60·2	0·25	1·40	5·1	

Tests Nos. 45 to 54 inclusive were made to compare the froth removing properties of the various Fumols. For this reason all tests were as nearly as possible of the same duration.

DIAGRAM IV

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 26

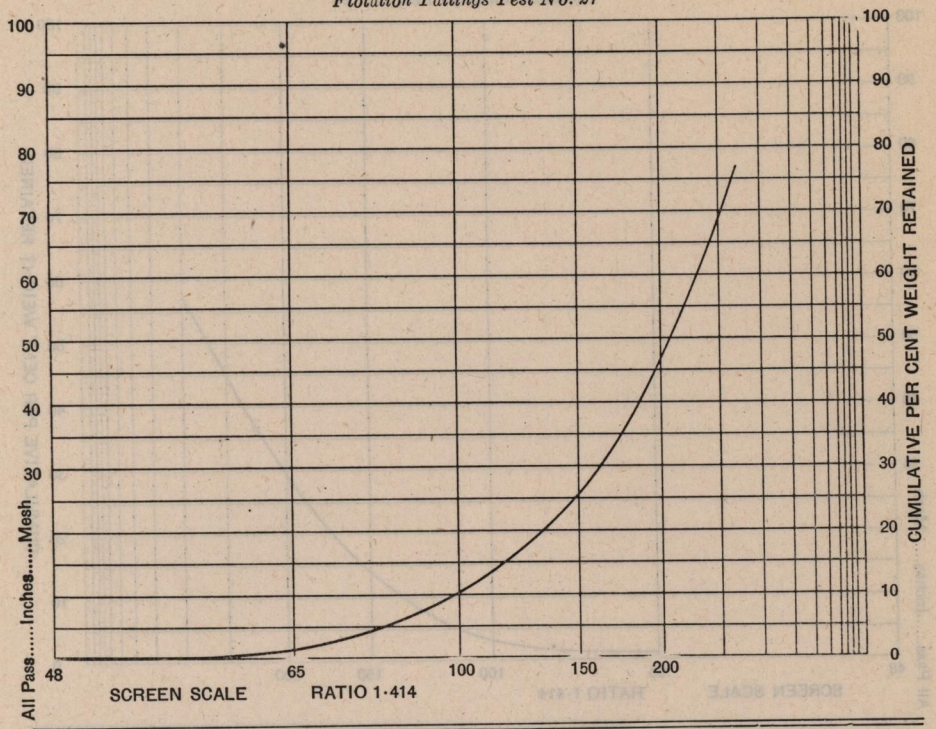


Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
.....	1·050	26·67	.....	·149	.....	.....	.....
.....	·742	18·85	.....	·135	.....	.....	.....
.....	·525	13·33	.....	·105	.....	.....	.....
.....	·371	9·423	.....	·092	.....	.....	.....
.....	·263	6·680	3	·070	.....	.....	.....
.....	·185	4·699	4	·065	.....	.....	.....
.....	·131	3·327	6	·036	.....	.....	.....
.....	·093	2·362	8	·032	.....	.....	.....
.....	·065	1·651	10	·035	.....	.....	.....
.....	·046	1·168	14	·025	.....	.....	.....
.....	·0328	·833	20	·0172	.....	.....	.....
.....	·0232	·589	28	·0125	.....	.....	.....
.....	·0164	·417	35	·0122	3·0	0·52	.....
.....	·0116	·295	48	·0092	14·0	2·43	2·95
.....	·0082	·208	65	·0072	39·0	6·76	25·13
.....	·0058	·147	100	·0042	89·0	15·42	39·51
.....	·0041	·104	150	·0026	83·0	14·38	55·29
Pass.....	·0029	·074	200	·0021	91·0	15·77	100·00
	·0029	·074	200	·0021	258·0	44·71	.....
Totals.....	.....	.....	.....	.....	577·0	.....	.....

DIAGRAM V

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 27

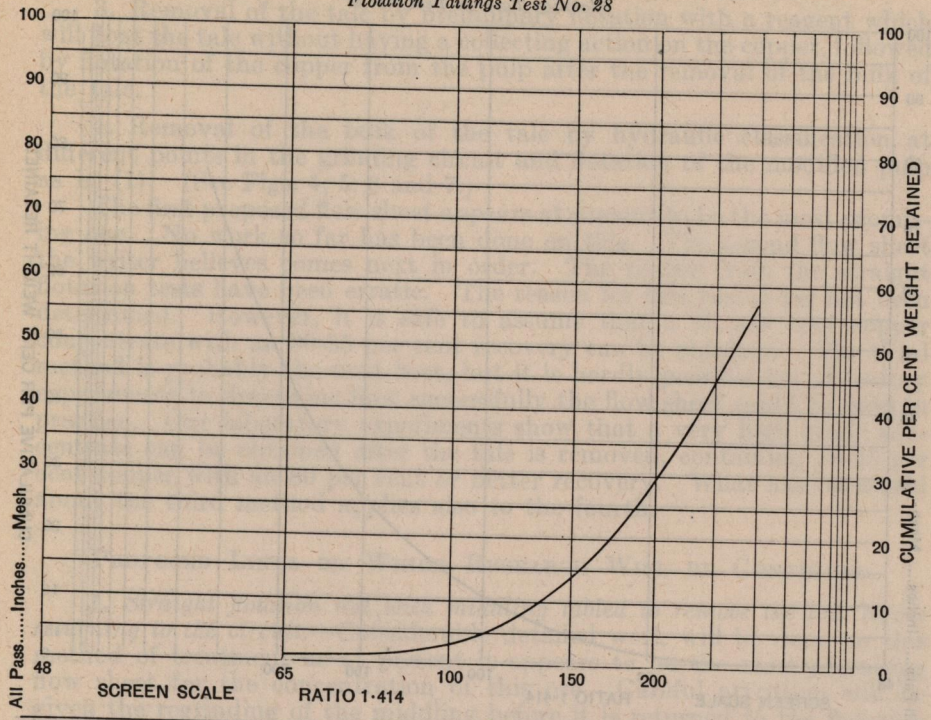


Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
.....	1·050	26·67	.....	·149	.....	.....	.....
.....	·742	18·85	.....	·135	.....	.....	.....
.....	·525	13·33	.....	·105	.....	.....	.....
.....	·371	9·423	.....	·092	.....	.....	.....
.....	·263	6·680	3	·070	.....	.....	.....
.....	·185	4·699	4	·065	.....	.....	.....
.....	·131	3·327	6	·036	.....	.....	.....
.....	·093	2·362	8	·032	.....	.....	.....
.....	·065	1·651	10	·035	.....	.....	.....
.....	·046	1·168	14	·025	.....	.....	.....
.....	·0328	·833	20	·0172	.....	.....	.....
.....	·0232	·589	28	·0125	.....	.....	.....
.....	·0164	·417	35	·0122	.....	.....	.....
.....	·0116	·295	48	·0092	3·0	0·49	.....
.....	·0082	·208	65	·0072	8·0	1·31	1·90
.....	·0058	·147	100	·0042	50·0	8·17	9·97
.....	·0041	·104	150	·0026	99·0	15·36	25·33
.....	·0029	·074	200	·0021	132·0	21·57	47·00
Pass.....	·0029	·074	200	·0021	325·0	53·10	100·00
Totals.....	.....	.....	.....	.....	612·0	100·00	.....

DIAGRAM VI

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 28

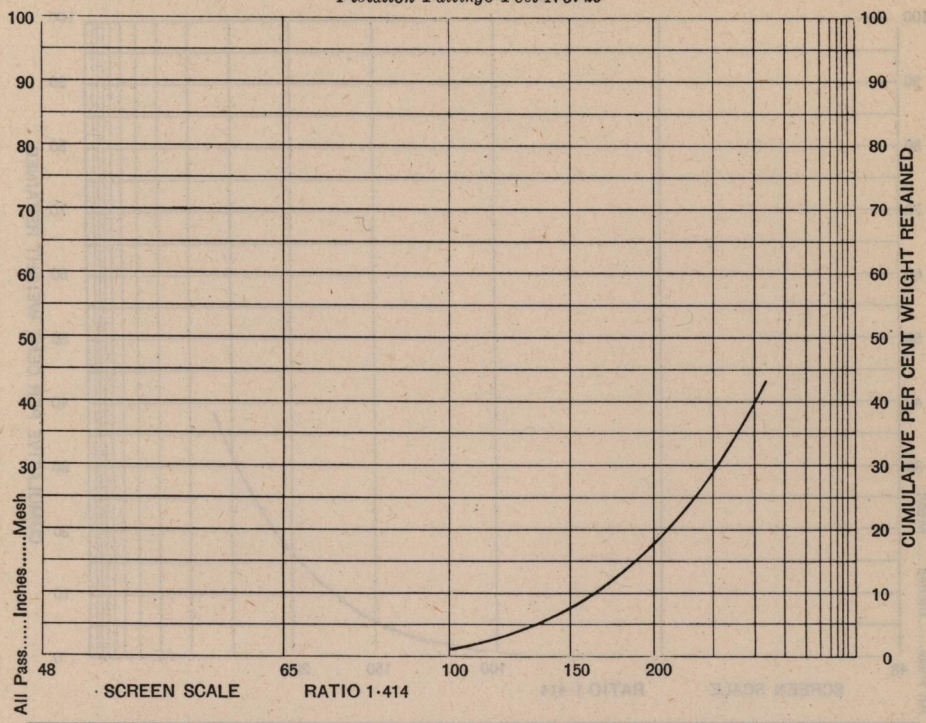


Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
	1·050	26·67	.....	·149	.....	.....	.....
	·742	18·85	.....	·135	.....	.....	.....
	·525	13·33	.....	·105	.....	.....	.....
	·371	9·423	.....	·092	.....	.....	.....
	·263	6·680	3	·070	.....	.....	.....
	·185	4·699	4	·065	.....	.....	.....
	·131	3·327	6	·036	.....	.....	.....
	·093	2·362	8	·032	.....	.....	.....
	·065	1·651	10	·035	.....	.....	.....
	·046	1·168	14	·025	.....	.....	.....
	·0328	·833	20	·0172	.....	.....	.....
	·0232	·589	28	·0125	.....	.....	.....
	·0164	·417	35	·0122	.....	.....	.....
	·0116	·295	48	·0092	.....	.....	.....
	·0082	·208	65	·0072	2·0	0·31	.....
	·0058	·147	100	·0042	14·0	2·17	2·49
	·0041	·104	150	·0026	72·0	11·23	13·73
Pass.....	·0029	·074	200	·0021	100·0	15·60	29·33
	·0029	·074	200	·0021	453·0	70·67	100·00
Totals.....	.....	.....	.....	.....	641·0	99·99	.....

DIAGRAM VII

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 29

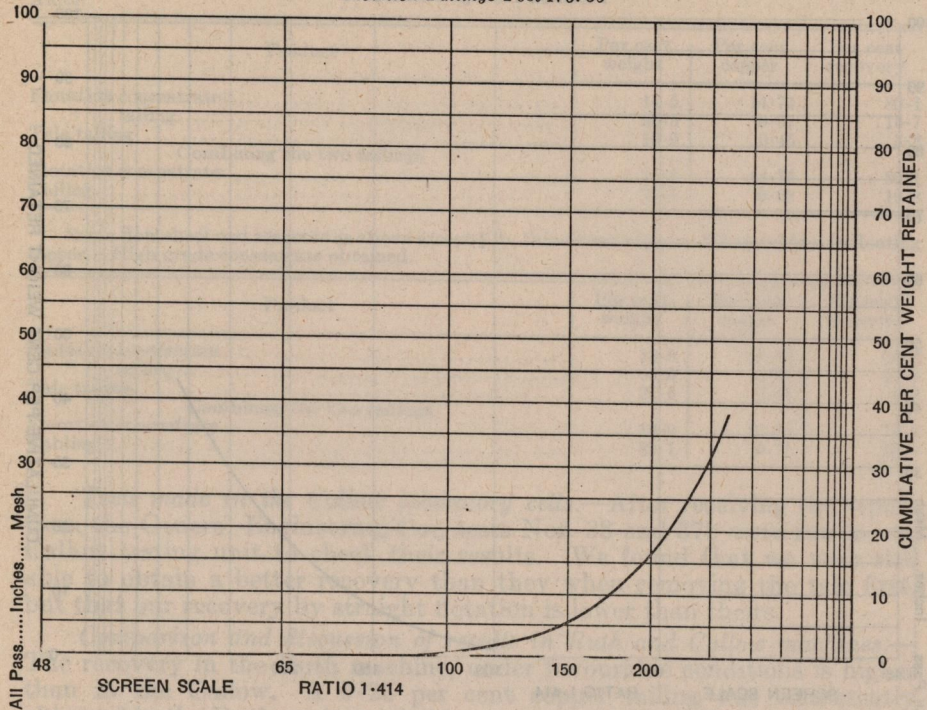


Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
.....	1·050	26·67	.....	·149	.....	.....	.....
.....	·742	18·85	.....	·135	.....	.....	.....
.....	·525	13·33	.....	·105	.....	.....	.....
.....	·371	9·423	.....	·092	.....	.....	.....
.....	·263	6·680	3	·070	.....	.....	.....
.....	·185	4·699	4	·065	.....	.....	.....
.....	·131	3·327	6	·036	.....	.....	.....
.....	·093	2·362	8	·032	.....	.....	.....
.....	·065	1·651	10	·035	.....	.....	.....
.....	·046	1·168	14	·025	.....	.....	.....
.....	·0328	·833	20	·0172	.....	.....	.....
.....	·0232	·589	28	·0125	.....	.....	.....
.....	·0164	·417	35	·0122	.....	.....	.....
.....	·0116	·295	48	·0092	.....	.....	.....
.....	·0082	·208	65	·0072	.....	.....	.....
.....	·0058	·147	100	·0042	7	1·18	1·18
.....	·0041	·104	150	·0026	41	6·79	7·95
.....	·0029	·074	200	·0021	60	9·93	17·88
Pass.....	·0029	·074	200	·0021	496	82·12	100·00
Totals.....	.....	.....	.....	.....	604	100·00	.....

DIAGRAM VIII

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 30



Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
.....	1·050	26·67	.....	·149	.....	.....	.....
.....	·742	18·85	.....	·135	.....	.....	.....
.....	·525	13·33	.....	·105	.....	.....	.....
.....	·371	9·423	.....	·092	.....	.....	.....
.....	·263	6·680	3	·070	.....	.....	.....
.....	·185	4·699	4	·065	.....	.....	.....
.....	·131	3·327	6	·036	.....	.....	.....
.....	·093	2·362	8	·032	.....	.....	.....
.....	·065	1·651	10	·035	.....	.....	.....
.....	·046	1·168	14	·025	.....	.....	.....
.....	·0328	·833	20	·0172	.....	.....	.....
.....	·0232	·589	28	·0125	.....	.....	.....
.....	·0164	·417	35	·0122	.....	.....	.....
.....	·0116	·295	48	·0092	.....	.....	.....
.....	·0082	·208	65	·0072	.....	.....	.....
.....	·0058	·147	100	·0042	2	·39	·39
.....	·0041	·104	150	·0026	25	4·95	5·35
.....	·0029	·074	200	·0021	58	11·51	16·86
Pass.....	·0029	·074	200	·0021	419	83·13	100·00
Totals.....	.....	.....	.....	.....	504	100·00	.....

Test No. 162

THE ELIMINATION OF THE IMPURITIES FROM MALAGASH ROCK SALT

R. K. Carnochan

Samples of rock salt from Malagash, N.S., were received at the Ore Dressing Laboratories upon the following dates:—

May 8, 1922,	Sample No. 1	5·5	pounds net.
" 22, 1922,	" 2	9·5	"
" 22, 1922,	" 3	2·5	"
" 22, 1922,	" 4	5·0	"
" 22, 1922,	" 5	5·5	"
June 26, 1922,	" 6	871·0	"

The rock salt was sent in by Messrs. Chambers and McKay, New Glasgow, N.S., and was discoloured. Tests were desired on this material to see if a process could be devised to remove the colour and produce a clean white salt.

The discoloration in the salt occurs in different ways. Some pieces are very dark and must contain a large amount of the discolouring matter, while others show just a faint even tint, and can only have a very small percentage of the impurity.

Analytical work by the chemists of the laboratories shows the colouring to be due to iron and organic matter. The salt as received ran about 7 per cent water insoluble, of which 1·8 per cent was Fe_2O_3 and 0·7 per cent organic matter. This shows that the percentage of colouring in the salt is very small.

The discoloration is the chief objection to the salt, but the insoluble is also undesirable, for when used to pack fish it remains on them as a slime.

The following methods of treatment were tried:—

1. Dissolving in water, filtering, and evaporating the filtrate. This gives a pure clean white salt. Before considering the installation of such a process, it was desired that experimental work be conducted to determine if the impurities could be removed by other methods involving a less costly installation.

2. The rock salt was sized and each size fed to a revolving furnace, temperature about 1100° F. The furnace discharge was sized. It was expected that either the pure salt or the discoloured salt might decrepitate and go into the fines when the furnace discharge was screened. It was found that both decrepitated about the same amount, for when screening no separation was evident.

3. Tabling the salt in a saturated salt solution on a small Wilfley table. This gives no concentration whatever as the discoloured particles seem to be of the same specific gravity as the pure salt.

4. Dry magnetic separation in the Ullrich machine. This process removes only the very dirty pieces of salt. If the sample which is fed to the machines is quite dirty, it makes some difference, but when the sample is only slightly discoloured it does not remove anything.

5. The salt was sized and each size ground in a small pebble mill. The mill discharge was screened. It was thought that the discoloured particles would not grind as fast as the pure salt particles, and upon screening the mill product it would be found in the coarse product. It was found that this method would not work as the discoloured material grinds almost as easily as the pure salt.

6. Washing the salt −80 mesh in a log washer using a concentrated salt solution as a wash. The washed salt was no cleaner than the feed to the machine.

7. Melting the salt and allowing the dirt to settle, while the salt is molten. This gives some separation—a lot of the dirt settles, but the salt produced is still discoloured.

CONCLUSIONS

None of the methods investigated would be suitable for the treatment of the salt for the elimination of the impurities, except that of dissolving and evaporating, the common practice with the majority of producers.

If this method is not adopted, closer selection of clean faces underground, closer sorting on the surface, with the rejection of the discoloured material, will be necessary to produce a clean salt in the various sizes required by the trade.

Test No. 163

THE CONCENTRATION OF GRAPHITE ORE, FROM THE TIMMINS MINE, WESTPORT, ONTARIO

C. S. Parsons

A sample of graphite ore was received at the Ore Testing Laboratories, March 15, 1922, from the Timmins mine, Westport, Ont. This sample was submitted by Mr. H. P. H. Brumell.

It consisted of material which would pass a 6 mesh screen. The graphite was of the flake variety in a gangue of comparatively soft material, showing the effects of considerable weathering.

The sample was found to contain on analysis 11·50 per cent carbon.

The object of the test was to produce a high grade graphite concentrate with a minimum reduction in the sizes of flake compared to that contained in the original sample. The sample was crushed in rolls to 10 mesh. A sample of 1,000 grams was cut out and reduced to 20 mesh in a ball mill. The separation was made by flotation and table concentration. Coal oil and pine oil were used, both oils being added to the ball mill.

After grinding, the pulp was floated in a Ruth flotation machine. A rougher concentrate and tailing was made. The rougher concentrate was fed directly to a laboratory Wilfley table, and a considerable quantity of clean sand was eliminated which had been carried over in mechanical suspension in the heavy froth of graphite. This table operation seems feasible, as a clean product is obtained representing 21 per cent by weight of the rougher concentrate. The elimination of the sand throws less work on the regrinding mills and the subsequent separation and flotation

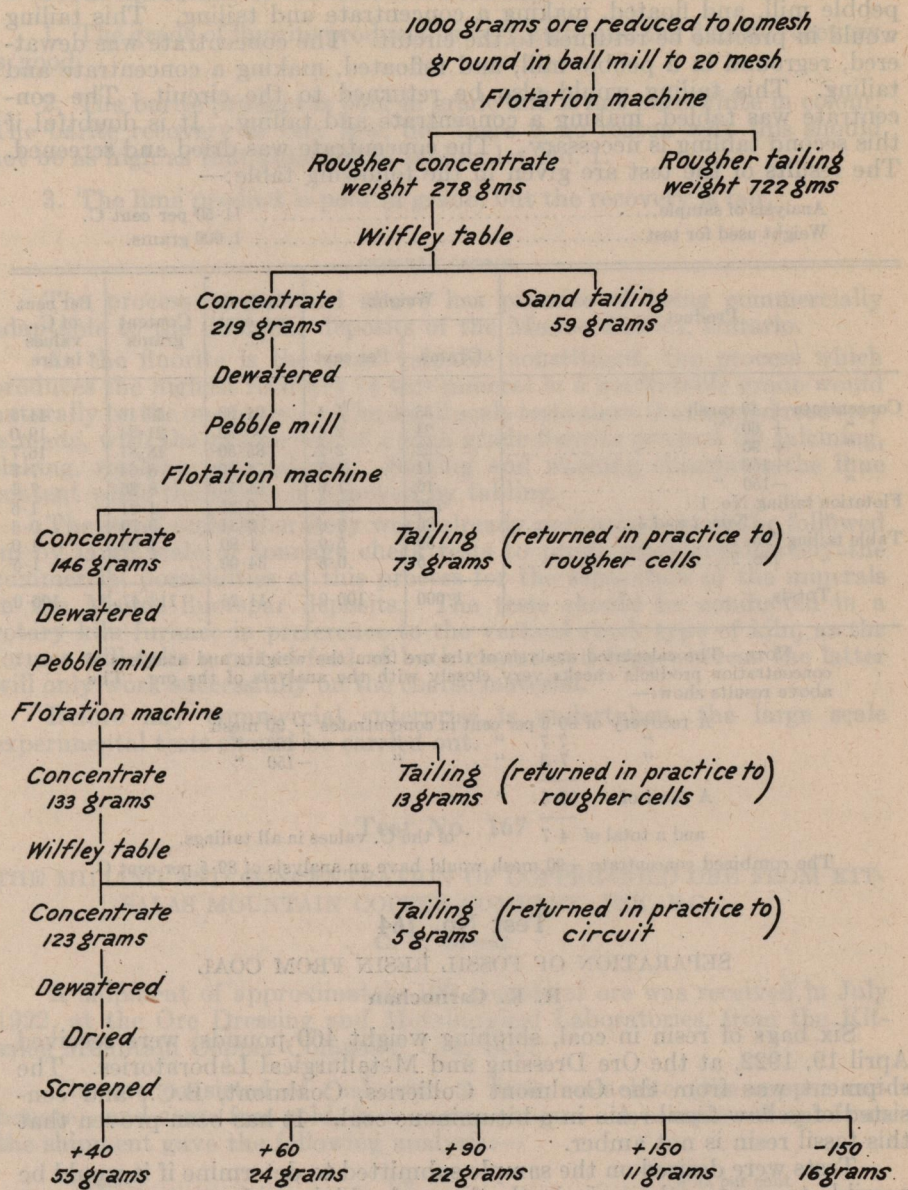


FIG. 8. Flow sheet of test made on 1,000 grams graphite ore from Timmins mine, Westport, Ontario.

machines. The concentrate from the table was dewatered, reground in a pebble mill, and floated, making a concentrate and tailing. This tailing would in practice be returned to the circuit. The concentrate was dewatered, reground in a pebble mill, and refloated, making a concentrate and tailing. This tailing would also be returned to the circuit. The concentrate was tabled, making a concentrate and tailing. It is doubtful if this second tabling is necessary. The concentrate was dried and screened. The results of the test are given in the following table:—

Analysis of sample..... 11·50 per cent C.
Weight used for test..... 1,000 grams.

Product	Weight		Analysis per cent C.	Content grams	Per cent of C.. values in ore
	Grams	Per cent			
Concentrate + 40 mesh.....	55	5·5	91·20	50·16	44·6
" + 60 "	24	2·4	88·90	21·34	19·0
" + 90 "	22	2·2	85·50	18·81	16·7
" +150 "	11	1·1	78·50	8·64	7·7
" −150 "	16	1·6	51·55	8·25	7·3
Flotation tailing No. 1.....	722	72·2	0·25	1·81	1·6
" Nos. 2 and 3.....	86	8·6	0·81	0·70	0·6
Table tailing No. 1.....	59	5·9	1·80	1·06	1·0
" No. 2.....	5	0·5	34·00	1·70	1·5
Totals.....	1,000	100·0	11·25	112·47	100·0

NOTE.—The calculated analysis of the ore from the weights and analyses of concentration products checks very closely with the analysis of the ore. The above results show:—

A recovery of 80·3 per cent in concentrates + 90 mesh
" 7·7 " " " +150 "
" 7·3 " " " −150 "

A total of 95·5 "

and a total of 4·7 " of the C. values in all tailings.

The combined concentrate +90 mesh would have an analysis of 89·5 per cent C.

Test No. 164

SEPARATION OF FOSSIL RESIN FROM COAL

R. K. Carnochan

Six bags of resin in coal, shipping weight 460 pounds, were received April 19, 1922, at the Ore Dressing and Metallurgical Laboratories. The shipment was from the Coalmont Collieries, Coalmont, B.C., and consisted of yellow fossil resin in a bituminous coal. It has been proven that this fossil resin is not amber.

Tests were desired on the sample submitted to determine if it would be possible to recover the resin in the form of a high grade concentrate, and leave a tailing of clean coal. The heavy solution method was used to separate the resin from the coal. In this process the resin is floated on the surface of an aqueous solution of sodium chloride or other salt, and the coal sinks in the same solution.

Six tests in all were made, five being on a small scale, and one on a large scale. It was found that the coal had to be crushed to pass 4 mesh before the resin was freed sufficiently.

Test No. 1.—This test was made in a tub. The coal and resin were thoroughly mixed with solution and then allowed to stand until the resin collected on top of the solution and the coal settled to the bottom. The resin with some solution was then poured off gently onto a 100 mesh screen. The solution from the screen was returned to the tub, the contents of the tub stirred, and then allowed to stand, after which the resin was again decanted from the tub. This decantation was made three times and it was found that this gave a clean tailing but a rather dirty concentrate. The concentrate was cleaned by putting it along with some solution in a tub, and decanting it once in the same manner as before. The middlings were put with the tails, and then both concentrates and tails were washed three times with fresh water to remove the salt solution, and then dried and weighed.

Solution used—Sodium chloride, density 1·15.
Size of feed, $-\frac{1}{4}''$

Product	Weight pounds	Per cent of heads
Concentrate	0·18	1·29
Tailing	13·82	98·71
Heads	14·00	100·00

Test No. 2, Run No. 1.—This test was made in a small tank. The coal and resin were mixed with the solution by means of a horizontal revolving cross-arm at the bottom of the tank the arm being driven by a vertical shaft. A spray of solution was also used to wet the coal which, when dry, floated on the surface of the solution in the tank. After mixing, the contents of the tank were allowed to stand a while and then more solution was poured slowly into the centre of the tank and the floating resin was allowed to overflow into a circular launder running around the outside of the tank. By means of this launder the amber was conveyed to a 100 mesh screen to remove the solution. After overflowing, the contents of the tank were stirred and overflowed again, this process being repeated in all three times. The concentrates were then recleaned by putting them in the tank and overflowing in the same manner as before. The middlings were put with the tailing, and the concentrate and tailing were washed three times with fresh water and then dried and weighed.

Solution used—Sodium chloride, density 1·15.
Size of feed, $-\frac{1}{4}''$

Product	Weight pounds	Per cent of heads
Concentrate	0·67	1·56
Tailing	42·33	98·44
Heads	43·00	100·00

Thirty pounds of the tailing was screened on 20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	29·00	80·55	79·30
Tailing −20	7·00	19·45	19·14
Heads	36·00	100·00	98·44

Test No. 2, Run No. 2.—15½ pounds of +20 tailing from run No. 1 were crushed to pass 20 mesh and treated in the same manner as described under run No. 1, the concentrates being re-run to clean them.

Solution used—Saturated sodium chloride, density 1·20.
Size of feed, −20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·16	1·03	0·82
Tailing	15·34	98·97	78·48
Feed	15·50	100·00	79·30

Concentrate run No. 1	1·56	per cent of heads.
" " 2	0·82	" "
Total concentrate	2·38	" "

Test No. 2, Run No. 3.—13½ pounds of +20 tailing from run No. 1 were crushed to pass 20 mesh and screened on 60.

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing −20 +60	10·50	77·78	61·68
Tailing −60	3·00	22·22	17·62
Feed	13·50	100·00	79·30

The 10½ pounds of −20 + 60 tailing were treated in the same manner as described under run No. 1, the concentrates being re-run to clean them.

Solution used—Saturated sodium chloride, density 1·20.
Size of feed, −20+60.

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·10	0·95	0·59
Tailing	10·40	99·05	61·09
Feed	10·50	100·00	61·68

Concentrate run No. 1	1·56	per cent of heads.
" " 3	0·59	" "
Total concentrate	2·15	" "

Test No. 3, Run No. 1.—20 pounds of heads $-\frac{1}{4}''$ were treated in the same manner as described under test No. 2, run No. 1, the concentrates being re-run to clean them.

Solution used—Calcium chloride, density 1·16.
Size of feed, $-\frac{1}{4}''$

Product	Weight pounds	Per cent of heads
Concentrate	0·31	1·55
Tailing	19·69	98·45
Heads	20·00	100·00

19¼ pounds of tailing from run No. 1 were screened on 20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	13·50	70·13	69·04
Tailing −20	5·75	29·87	29·41
Feed	19·25	100·00	98·45

Test No. 3, Run No. 2.—13½ pounds of +20 tailing from run No. 1, were crushed to pass 20 mesh, and treated as described under test No. 2, run 1, the concentrate being re-run to clean them.

Solution used—Calcium chloride, density 1·16.
Size of feed, −20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·10	0·74	0·51
Tailing	13·40	99·26	68·53
Feed	13·50	100·00	69·04

Concentrate run No. 1	1·55	per cent of heads.
" " 2	0·51	" "
Total concentrate	2·06	" "

Test No. 4.—This test was made in exactly the same manner as test No. 3.

Run No. 1

Product	Weight pounds	Per cent of heads
Concentrate	0·30	1·50
Tailing	19·70	98·50
Heads	20·00	100·00

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	13·20	67·00	66·00
Tailing −20	6·50	33·00	32·50
Feed	19·70	100·00	98·50

Run No. 2

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·08	0·60	0·41
Tailing	12·92	99·40	65·59
Feed	13·00	100·00	66·00

Concentrate run No. 1 ..... 1·50 per cent of heads.
" " 2 ..... 0·41 " "
Total concentrate ..... 1·91 " "

Test No. 5.—This test was made in exactly the same manner as test No 3.

Run No. 1

Product	Weight pounds	Per cent of heads
Concentrate	0·30	1·52
Tailing	19·50	98·48
Heads	19·80	100·00

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	16·00	82·05	80·80
Tailing −20	3·50	17·95	17·68
Feed	19·50	100·00	98·48

Run No. 2

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·06	0·40	0·32
Tailing	14·94	99·60	80·48
Feed	15·00	100·00	80·80

Concentrate run No. 1 ..... 1·52 per cent of heads.
" " 2 ..... 0·32 " "
Total concentrate ..... 1·84 " "

Test No. 6.—This test was made on 303½ pounds of heads −20 mesh. A rough concentrate was made by decanting from a tub as described under test No. 1, about one-twelfth of the lot being treated at a time. The rough concentrate was mixed with solution in a large cone and allowed to stand. Any coal that settled out was drawn off from the bottom, and the contents of the cone were then stirred and allowed to settle and the coal drawn out from the bottom again. This procedure was continued until only the resin concentrate remained in the cone. The coal from the bottom of the cone was put into the cone again and rerun in the same manner to get any resin that might be in it. All products were washed three times in fresh water and dried and weighed. In washing the two concentrates produced, the wash water was decanted into a filter press and this gave a small amount of very fine concentrate which was kept separately.

Solution used—Sodium chloride, density 1·17.
Size of feed, −20 mesh.

Product	Weight pounds	Per cent of heads
1st concentrate (from cone)	6·50	2·14
2nd concentrate (from cone)	0·50	0·16
3rd concentrate (from press)	0·24	0·08
1st tailing (from tub)	230·00	75·78
2nd tailing (from cone)	57·50	18·95
Loss	8·76	2·89
Heads	303·50	100·00

Total concentrate..... 2·38 per cent of heads.

SUMMARY

Test No.	No. 1 conc. per cent heads	No. 2 conc. per cent heads	No. 3 conc. per cent heads	Total conc. per cent heads	Soln. used	Density first treatment	Density second treatment	Feed first treatment	Feed second treatment
1	1·29	.....	.....	1·29	NaCl	1·15	.....	−¼″	.....
2, run Nos. 1 and 2	1·56	0·82	.....	2·38	NaCl	1·15	1·20	−¼″	−20
2, run Nos. 1 and 3	1·56	0·59	.....	2·15	NaCl	1·15	1·20	−¼″	−20+60
3	1·55	0·51	.....	2·06	CaCl	1·16	1·16	−¼″	−20
4	1·50	0·41	.....	1·91	CaCl	1·16	1·16	−¼″	−20
5	1·52	0·32	.....	1·84	CaCl	1·16	1·16	−¼″	−20
6	2·14	0·16	0·08	2·38	NaCl	1·17	1·17	−20	−20

CONCLUSIONS

1. It is possible to recover the resin from the coal. Although no analyses on the heads and products have been obtained, it is evident from a close inspection, that the concentrates are high grade, and the tails clean. This indicates a good recovery of the resin in a suitable form.

2. Sodium chloride is the best salt to make up the solution used in the separation, as it gives just as good results as calcium chloride, and is cheaper to use, in spite of the fact that more of it is needed to give a solution of a certain density.

3. The coal and resin should be ground to pass 20 mesh before separating, as this would give a much simpler flow sheet, use less equipment and power, and give just as good a product as separating at $-\frac{1}{4}''$, and regrinding and separating again.

4. A cone would be a suitable apparatus to use for separating, as it was found in the test work to work very nicely, and give the best separation.

5. In mixing the coal and resin with solution, some vigorous agitator will have to be used, as the coal seems to wet with difficulty, and when dry it floats on the surface of the solution.

Test No. 165

THE WET SEPARATION OF ASBESTOS FROM ITS GANGUE

R. K. Carnochan

In 1921 the Division of Ore Dressing and Metallurgy conducted dry milling tests on shipments of asbestos rock from two mines in the Quebec asbestos field. On the completion of these tests it was decided to investigate the wet milling of asbestos. This investigation was carried out as far as the means at the disposal of the division would allow. The asbestos rock with which the tests were made was the unused portion of a shipment received for dry milling from the Black Lake Asbestos and Chrome Company.

In all thirteen tests were made on this shipment, three large and one small scale dry milling tests, and nine small scale wet milling tests. To give an idea of the results obtained by wet milling, the last test will be given in detail.

Test No. 13

Run No. 1 (1st wet ball milling).—This test was made on 25 pounds of a head sample taken from the five tons of asbestos rock which was used in the first large scale dry milling test. The 25 pounds were crushed to pass $\frac{3}{4}''$ and fed to a small wet ball mill, the mill discharge was screened on $\frac{1}{2}''$, and the $-\frac{1}{2}''$ went to a small Richards pulsating jig. The jig separated the fibre from the rock. The fibre from the jig was run to a 4 mesh screen and the -4 mesh fell onto a 20 mesh screen. This gave a $+4$ mesh clean fibre and a $+20$ mesh fibre which contained some rock. The -20 fell into a tub, the overflow from the tub running to a series of two settling boxes. This gave a -20 sand product which contained some

fibre and two float products from the settling boxes which are composed of very fine fibre. After the 25 pounds had been fed the $+\frac{1}{2}''$ was dried and crushed to $-\frac{1}{2}''$ and fed back to the mill.

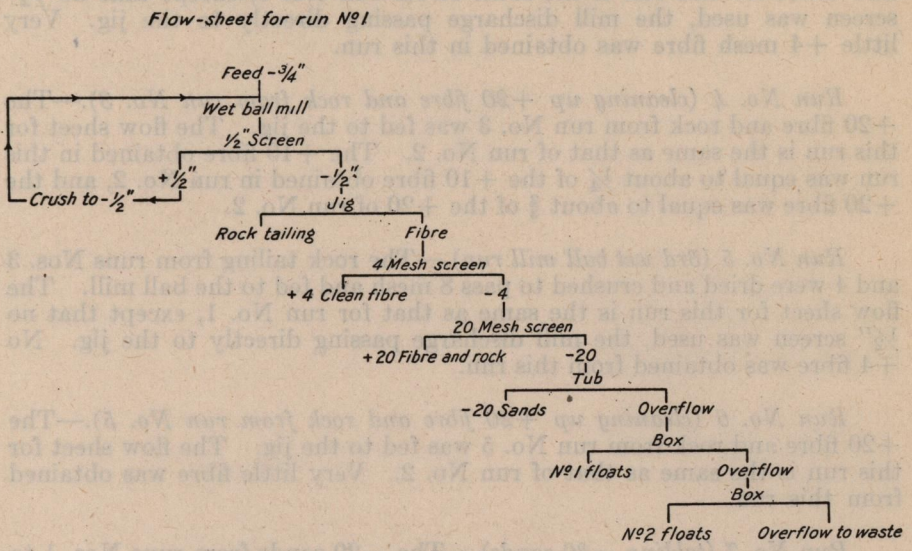


FIG. 9

Run No. 2 (cleaning up +20 fibre and rock from run No. 1).—The +20 fibre and rock from run No. 1 was fed to the jig making a rock tailing and a fibre product. The fibre was run to 10 and 20 mesh screens, giving a +10 clean fibre and a −20 clean fibre. The −20 was treated in the same way as the −20 of run No. 1. Flow sheet of run No. 2:—

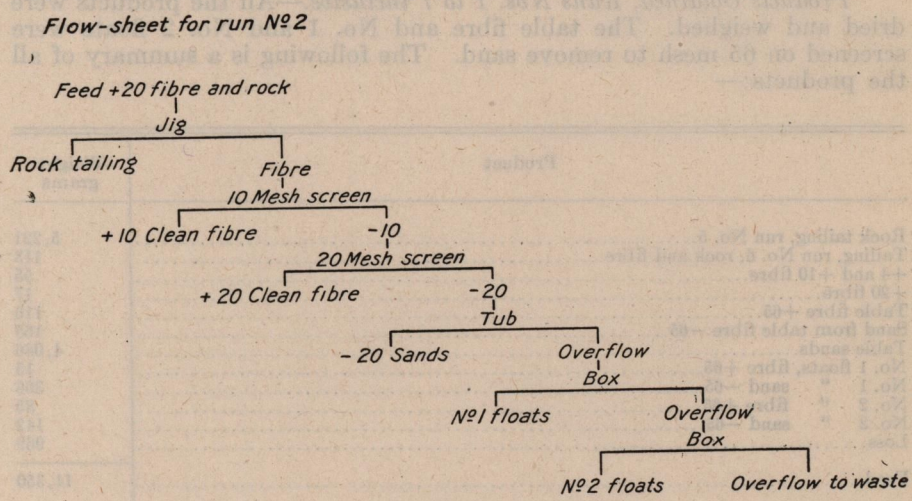


FIG. 10

Run No. 3 (2nd wet ball mill run).—The rock tailings from runs Nos. 1 and 2 were dried and crushed to pass 4 mesh and fed to the mill. The flow sheet for this run is the same as for run No. 1, except that no ½" screen was used, the mill discharge passing directly to the jig. Very little +4 mesh fibre was obtained in this run.

Run No. 4 (cleaning up +20 fibre and rock from run No. 3).—The +20 fibre and rock from run No. 3 was fed to the jig. The flow sheet for this run is the same as that of run No. 2. The +10 fibre obtained in this run was equal to about ¼ of the +10 fibre obtained in run No. 2, and the +20 fibre was equal to about ⅔ of the +20 of run No. 2.

Run No. 5 (3rd wet ball mill run).—The rock tailing from runs Nos. 3 and 4 were dried and crushed to pass 8 mesh and fed to the ball mill. The flow sheet for this run is the same as that for run No. 1, except that no ½" screen was used, the mill discharge passing directly to the jig. No +4 fibre was obtained from this run.

Run No. 6 (cleaning up +20 fibre and rock from run No. 5).—The +20 fibre and rock from run No. 5 was fed to the jig. The flow sheet for this run is the same as that of run No. 2. Very little fibre was obtained from this run.

Run No. 7 (tabling −20 sands).—The −20 sands from runs Nos. 1 to 6 inclusively were tabled on a small Wilfley table making a sand product, a middling, and a fibre product. The middling was re-run twice, and in the second re-run only two products were made. The overflow from the boxes used to collect the table products was led to the tub and two settling boxes that were used in the other runs.

Products Obtained, Runs Nos. 1 to 7 inclusive.—All the products were dried and weighed. The table fibre and No. 1 and No. 2 floats were screened on 65 mesh to remove sand. The following is a summary of all the products:—

Product	Weight grams
Rock tailing, run No. 5	5,221
Tailing, run No. 6, rock and fibre	113
+4 and +10 fibre	55
+20 fibre	17
Table fibre +65	116
Sand from table fibre −65	165
Table sands	4,086
No. 1 floats, fibre +65	15
No. 1 " sand −65	366
No. 2 " fibre +65	85
No. 2 " sand −65	142
Loss	969
Feed	11,350

Fibre Obtained, Runs Nos. 1 to 7 inclusive.—All the fibre obtained was rubbed between the hands to loosen it up as in drying it sticks together. Each lot of fibre was then tested on a standard testing machine with the following results:—

+4 and +10 fibre.....	+ 2.....	3	grams.		
	+ 4.....	27	"		
	+10.....	14	"		
	+10.....	10	"		
+20 fibre.....	+ 2.....	0	"		
	+ 4.....	2	"		
	+10.....	5	"		
	−10.....	10	"		
Table fibre.....	+ 2.....	0	"		
	+ 4.....	0	"		
	+10.....	22	"		
	−10.....	95	"		
No. 1 float fibre.....	+ 2.....	0	"		
	+ 4.....	0	"		
	+10.....	4	"		
	−10.....	11	"		
No. 2 float fibre.....	+ 2.....	0	"		
	+ 4.....	5	"		
	+10.....	33	"		
	−10.....	46	"		
Total fibre.....	+ 2.....	3	"	0·026	per cent of heads.
	+ 4.....	34	"	0·300	" "
	+10.....	78	"	0·687	" "
	−10.....	172	"	1·515	" "
		287	"	2·528	" "

CONCLUSIONS

1. As the test was on a small scale the results will only roughly approximate those that would be obtained in large scale work. In a large ball mill the fibre would be cut up more before being discharged and hence less coarse fibre would be obtained. In testing the fibre in the standard testing machine a pound of fibre should have been used, but as none of the fibre products weighed a pound, the tests had to be made on a much smaller amount. The result of this would be that the coarse fibre would have a better opportunity to work through the screens and the tests would indicate a lower grade of fibre than that really obtained. In the work on dry milling of asbestos rock it was found that large and small scale tests on similar feeds did not check. This is due to the great difficulty of getting a small representative sample.

2. The following table shows the total fibre obtained in test No. 13 compared with that obtained in the first large dry milling test. Both these tests were on similar rock, the feed to test No. 13 being part of a head sample cut out of the feed of the large dry milling test.

	Total fibre. Test 13	Per cent of heads. Dry milling test
+ 2	0·026	0·048
+ 4	0·300	0·265
+10	0·687	0·346
−10	1·515	0·430
Total	2·528	1·089

The results of the wet milling test compare favourably with those of the dry milling test. As to which is the best milling method, it would be difficult to say with the present data. This question could only be decided by comparative large scale tests which it is hoped the Division of Ore Dressing and Metallurgy will be able to carry out in the near future.

Test No. 166

THE SEPARATION OF FLUORITE, CALCITE, AND BARITE IN THE FLUORSPAR FROM THE MADOC DISTRICT, ONT.

R. K. Carnochan

Three shipments of barite, fluorite, and calcite were received at the Ore Dressing and Metallurgical Laboratories from Gavin M. Wallbridge, manager of the Wallbridge mine, Madoc, Ont. These shipments were obtained from a vein on the west half of lot 1, con. I, Madoc township, Hastings county, Ont.

Shipment	Date received	Net weight	$BaSO_4$ per cent	CaF_2 per cent	$CaCO_3$ per cent	SiO_2 per cent
		pounds				
Lot No. 1	Feb. 12, 1919	287	4·40	48·35	40·00	3·10
Lot No. 2	April 3, 1922	505	6·00	72·80	15·60	1·55
Lot No. 3	June 16, 1922	1,368	6·00	76·40	15·50	.....

The three lots consisted of a natural mixture of barite, fluorite and calcite, the size of the individual pieces of mineral ranging from about 2 inches to very small grains. Tests were desired to determine if it would be possible to separate the three minerals into products pure enough to be marketable.

Tabling

Lot No. 1 was used in making table tests. A full report covering these tests was published in the Summary Report of the Mines Branch for 1919 (see test No. 115, page 71).

The conclusions derived from these table tests were:—

1. To secure good work by tabling, the feed will have to be ground to −80 mesh.

2. The slime and dust loss in the tests made on −80 mesh material was very high, but with better equipment this loss could be lowered.

3. If the loss could be kept down to 10 per cent, the following grades of products and recoveries would be obtained by tabling at −80 mesh:—

Product	Recovery per cent	Grade per cent
Barite	75·5	81·85
Fluorite	65·2	74·80
Calcite	63·7	59·20

4. Lot No. 1 is lower in barite and fluorite than either lots No. 2 or No. 3. Better barite and fluorite products, and better recoveries of barite and fluorite would have been obtained by tabling material similar to lot No. 2 or lot No. 3.

Decrepitation

The Rock Candy Fluorspar Mill, Grand Forks, B.C., is successfully concentrating a fluorite by decrepitation. The fluorite is crushed, dried, and sized. Each size, excepting the fines, is fed into a separate rotary kiln, temperature 1200°F. In passing through the kiln, the fluorite decrepitates while the gangue does not, and the discharges from the different kilns are screened on suitable screens, the throughs being finished concentrate and the oversize tailings. The fines from the first sizing are concentrated by tabling.

This decrepitation process was tried on part of lot No. 2, a portion of which was crushed to 4 mesh and then carefully sized down to 20 mesh. Each size was fed separately to a small revolving furnace heated to about 1200°F. Upon entering the furnace the fluorite was seen to glow and then decrepitate. The furnace discharge from each size fed to the furnace was screened into a suitable number of sizes and each of these sizes was analysed for fluorite. It was found that the different sizes all ran about the same in fluorite, the fines in most cases running a little higher than the coarse. Decrepitation tests made by placing small pieces of fluorite and calcite picked from lot No. 2 on a heated wire gauze, show that the calcite when crystalline decrepitates just as readily as the fluorite, but when the calcite is not crystalline, it does not decrepitate. This explains the poor results obtained by decrepitation.

The conclusion derived from these decrepitation tests was that the method is unsuitable for the concentration of the fluorite in the Madoc deposits.

Jigging

A large scale jig test was made on 381 pounds of lot No. 2 −4 mesh. The products of this test, after sampling for analysis, were mixed together and used to make another large scale jig test.

Analyses on the products of these jig tests show very little separation of the minerals.

From these tests it was concluded that jigging was not a suitable method for use in treating the material submitted.

Calcining, Slacking, Washing and Tabling

A method of removing lime from magnesite had been developed by the Mines Branch when working on magnesite from Grenville township, Que. This method consists of calcining the magnesite, slacking in boiling water, upon which the lime goes into a thin slurry while the magnesia stays coarse, and then washing out the lime.

It was decided to try and adapt this process to remove the calcite from the barite and fluorite, with the subsequent separation of the barite and fluorite by tabling.

Two small scale tests were made on lot No. 2, and one medium scale test on lot No. 3. These tests gave similar results.

In the medium scale test, 29 pounds of lot No. 3 -4 mesh were calcined at 1400°F. The calcined material ran 0·70 per cent CO_2 and weighed 27 pounds. This 27 pounds was sampled to 26·5 pounds, slacked in boiling water, and washed by agitating in pails and decanting. The washed material when dried weighed 20 pounds and analysed $BaSO_4$ 5·9 per cent, CaF_2 87·4 per cent, CaO 2·38 per cent. The washings weighed 6·5 pounds and analysed $BaSO_4$ 8·7 per cent, CaF_2 51·3 per cent, CaO 26·4 per cent. A 9·75 pound sample of the washed material was crushed to pass 40 mesh and tabled.

Table concentrate	0·29 pounds	$BaSO_4$	52·4 per cent.
Table tailing	8·75 "	CaF_2	91·7 "
Table slime	0·17 "	CaF_2	71·5 "

A 10 pound portion of washed material was crushed to pass 80 mesh and tabled.

Table concentrate	0·26 pounds	$BaSO_4$	52·5 per cent.
Table tailing	8·50 "	CaF_2	93·5 "
Table slime	0·52 "	CaF_2	86·4 "

Tabling at −40 mesh gives the following recoveries and grades of products:—

Product	Recovery per cent	Grade per cent
Barite—table concentrate	18·3	52·4 $BaSO_4$
Fluorite—table tail and slime	76·8	91·3 CaF_2
Lime—washings	69·4	26·4 CaO

Tabling at −80 mesh gives the following recoveries and grades of products:—

Product	Recovery per cent	Grade per cent
Barite—table concentrate	16·0	52·5 $BaSO_4$
Fluorite—table tail and slime	77·2	93·1 CaF_2
Lime—washings	69·4	26·4 CaO

From the results of the tests it is seen that—

1. The grade of fluorite produced is very high, and the fluorite recovery is good.

2. The barite product is poor in grade and is not pure white in colour. The barite recovery is very low, but there is no reason why this should not be as high as that obtained in tabling lot No. 1.

3. The lime product is poor in grade, but the recovery is fair.

CONCLUSIONS

The process as outlined above has promise of being commercially adaptable to the fluorspar deposits of the Madoc district, Ontario.

As the fluorite is the most valuable constituent, the process which produces the highest recovery of this mineral in a marketable grade would naturally be the one to use. The small scale tests show that a good recovery is made, with the production of a high grade fluorite product by calcining, slaking, washing, and tabling. Slaking and washing eliminates the lime content while the barite is removed by tabling.

The small scale laboratory work already performed should be followed up by large scale or tonnage check tests to prove more conclusively the commercial possibilities of this process for the separation of the minerals in the Madoc fluorspar deposits. The tests should be conducted in a rotary kiln furnace in preference to the vertical stack type of kiln, as the former will take a mixed feed of both coarse and fine, whereas the latter will only work successfully on the coarse material.

Before any commercial enterprise is undertaken, the large scale experimental tests should be carried out.

Test No. 167

THE MILLING AND CONCENTRATION OF COPPER-GOLD ORE FROM KITSALAS MOUNTAIN COPPER COMPANY, USK, B.C.

C. L. Dewar

A shipment of approximately 100 pounds of ore was received in July 1922, at the Ore Dressing and Metallurgical Laboratories, from the Kitsalas Mountain Copper Company, Usk, B.C.

The ore consisted of chalcocite, a little malachite (the copper carbonate), and some free gold, in a quartz gangue. A sample cut out from the shipment gave the following analysis:—

Copper	Cu	5·92 per cent
Iron	Fe	2·15 "
Sulphur	S	1·50 "
Silica	SiO_2	81·75 "
Lead	Pb	Trace.
Zinc	Zn	"
Gold	Au	0·46 oz. per ton.
Silver	Ag	2·17 " "

The company has a small mill on the property consisting of 8 x 10 Blake crusher, Gibson mill, amalgamation plates, and Wilfley concentrator. The ore is crushed in the Blake crusher, ground to 25 mesh in the Gibson mill with mercury, run over 5′ x 6′ amalgam plate, and concentrated on a Wilfley table. The concentrate has an assay value of—

Copper	Cu	26·88 per cent.
Gold	Au	0·65 oz.
Silver	Ag	16·45 oz.

By this treatment the free gold is amalgamated and the copper, silver, and remaining gold values recovered in the form of a copper concentrate.

Tests were conducted to determine methods of treatment by which a high recovery of the values in the ore could be obtained, with the production of a high grade copper concentrate which could be readily disposed of to the copper smelters.

Test No. 1.—Concentration by flotation.—1,000 grams of the ore previously crushed to 18 mesh were ground wet in a laboratory ball mill for 20 minutes, with 0·8 pound per ton of oil mixture—coal tar creosote 60 per cent, coal tar 40 per cent—and 0·2 pound per ton of Barrett No. 4. It was then floated in a laboratory Ruth flotation machine with the addition of 0·1 pound per ton frothing oil—P. T. and T. Co.'s No. 350. A concentrate and tailing were made.

86·9 per cent of the copper values, 82·1 per cent of the silver, but only 54·2 per cent of the gold values were recovered in a concentrate assaying 46·60 per cent copper, 17·75 ozs. silver, and 2·30 ozs. gold. The results of this test are shown in the table following, as is also the degree of crushing in table of screen analyses.

Test No. 2.—Table concentration followed by flotation of the table tailing.—1,500 grams of the ore previously crushed to 18 mesh were concentrated on a laboratory Wilfley table, making a concentrate, a middling, and a tailing. The middling product was re-run over the table, the concentrate from which was added to the first concentrate, and the tailing to the first tailing. As the pulp was passing over the table some small flakes of free gold (about 1 mm. diameter) were noticed. 51·0 per cent of the copper, 51·6 per cent of the gold, and 50·8 per cent of the silver was recovered in a concentrate assaying 53·80 per cent copper, 3·90 ozs. gold, and 20·60 ozs. silver.

The table tailing was reground for 40 minutes with 1 pound per ton of oil mixture—hardwood oil 50 per cent, wood tar 50 per cent—and floated in a laboratory Ruth machine. This further concentration of the values recovered 33·8 per cent of the copper, 23·0 per cent of the gold, and 33·6 per cent of the silver, in a concentrate assaying 34·45 per cent copper, 1·68 ozs. gold, 13·20 ozs. silver.

The combined table and flotation concentrates would give the following assay values and recoveries:—

Ratio of concentration—12 per cent of original feed, or 1 : 8·33. Assays—copper 43·9 per cent; gold, 2·77 ozs.; silver, 16·85 ozs. Recoveries—copper 84·8 per cent; gold 74·6 per cent; silver 84·4 per cent.

Test No. 3.—Amalgamation followed by flotation.—1,000 grams of the ore previously crushed to 18 mesh were ground wet for 30 minutes in a small ball mill, using a small ball charge, to which was added 100 grams of mercury. The mercury was recovered by panning the pulp away from it, and the pulp was reground for 15 minutes with 0·8 pound per ton coal tar mixture and 0·2 pound per ton Barrett No. 4. It was then floated in a laboratory Ruth flotation machine.

—	Copper per cent	Gold per cent	Silver per cent
Recoveries by amalgamation	.....	32·8	5·7
Recoveries in flotation concentrate	80·4	36·6	74·7
Total recoveries	80·4	69·4	80·4
		oz.	oz.
Grade of flotation concentrate	28·8	1·07	10·34

Test No. 4.—Flotation followed by table concentration.—1,000 grams of the ore previously crushed to 18 mesh were ground wet for 15 minutes with 0·8 pound per ton coal tar mixture and 0·2 pound per ton Barrett No. 4, and floated in a laboratory Ruth flotation machine. The tailing was dried, weighed and sampled, the remainder being tabled on a laboratory Wilfley table.

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs. Au	Ozs. Ag	Per cent Cu	Per cent Au	Per cent Ag
Flot. conc.	10·6	40·30	1·85	14·40	75·1	46·7	70·8
Table conc.	3·3	25·35	3·95	11·60	14·5	38·2	18·7
Combined conc.	13·9	36·50	2·35	13·75	89·6	84·9	89·5

A screen analysis showing the degree of crushing is given in the table following:—

Test No. 5.—Table concentration followed by flotation.—1,000 grams of the ore, previously crushed to 18 mesh was ground for 15 minutes and tabled on a laboratory Wilfley table, making a concentrate and a tailing. The tailing was floated in a laboratory Janney machine with 0·87 pound per ton coal tar mixture, 0·22 pound per ton Barrett No. 4, 0·11 pound per ton P. T. and T. Co.'s No. 350. The concentrate was recleaned in a laboratory Ruth machine with the addition of 0·1 pound per ton No. 5 pine oil.

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs-Au	Ozs-Ag	Per cent Cu	Per cent Au	Per cent Ag
Table conc.	8·8	49·33	5·25	18·80	72·4	83·3	71·3
Flot. conc.	2·1	39·80	2·40	17·00	14·0	9·1	15·4
Combined conc.	10·9	47·40	4·72	18·40	86·4	92·4	86·7

These recoveries do not include what would be recovered by the retreatment of the middling product, which would be returned to the circuit in practice. A screen analysis showing the degree of crushing, is given in the table following.

Test No. 6.—Table concentration followed by flotation.—1,200 grams of the ore, previously crushed to 24 mesh was tabled on a laboratory Wilfley table, making a concentrate, middling and tailing. The middling and tailing products were ground for 10 minutes with 0·9 pound per ton coal tar mixture, 0·2 pound per ton Barrett No. 4, and floated in a laboratory Ruth machine with the addition of 0·1 pound per ton P. T. and T. Co.'s No 350. A concentrate, middling, and tailing were made:—

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs-Au	Ozs-Ag	Per cent Cu	Per cent Au	Per cent Ag
Table conc.	8·9	48·95	3·06	23·94	74·0	72·4	78·5
Flot. conc.	3·4	26·33	2·38	11·78	14·8	20·7	14·5
Combined conc.	12·3	42·85	2·86	20·60	88·8	93·1	93·0

By retreating the flotation middling product, the recoveries could be increased slightly. The high copper tailing obtained in this test, namely 0·5 per cent Cu., and in the preceding one, 0·55 per cent Cu. is chiefly due to the presence of copper carbonate in the ore.

Test No. 7.—Table concentration of sized material followed by flotation.—While the recoveries obtained in tests Nos. 5 and 6 were as high as could be expected, test No. 7 was run to determine if the grade of the concentrate could be improved by sizing before table concentration, and thus produce a more suitable shipping product.

4,300 grams of the ore previously crushed to 18 mesh was sized on 35 and 70 mesh screens, as follows:—

−18+35	1,975 grams
−35+70	1,135 "
−70	1,190 "
Total	4,300 grams

The −18 +35 mesh material was tabled on a laboratory Wilfley table, making a concentrate, middling, and tailing. A clean concentrate was produced, but the material was a little coarse to obtain a clean tailing.

The −35 +70 mesh material was tabled in the same manner and the middlings combined. The amount of middling from the two lots was about the same, 13·7 per cent of the weight of the −18 + 35 mesh material and 14·5 per cent of the weight of the −35 +70 mesh material.

The −70 mesh material was ground in a ball mill for 10 minutes with 2 pounds per ton coal tar mixture and floated in a laboratory Ruth machine, making a concentrate and tailing.

The middling products from the table concentration of the −18 +35 and the −35 +70 mesh lots were reground and floated, using 1·8 pounds per ton of coal tar mixture, making a concentrate and tailing. The tailing products from the table concentration of the −18 +35 and the −35 +70 mesh lots were reground and floated, making a concentrate and tailing. A summary of the concentrates, assay values, and recoveries made in this test follows:—

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs-Au	Ozs-Ag	Per cent Cu	Per cent Au	Per cent Ag
Table conc. −18+35..	3·07	61·14	4·84	22·86	30·42	34·15	30·96
Table conc. −35+70..	2·23	60·50	4·90	23·12	21·85	25·08	22·71
Flot. conc. −70.....	3·44	45·53	1·56	14·08	25·40	12·30	21·41
Flot. conc. (re-run middlings).....	0·60	60·44	6·57	23·50	5·82	9·02	6·70
Combined conc.....	9·34	55·20	3·76	19·70	83·50	80·60	81·80
Flot. conc. (re-run tailing).....	1·19	22·82	3·67	15·75	4·23	9·93	8·40
Combined conc.....	10·53	51·50	3·75	19·25	87·70	90·50	90·20

If the middling products from table concentration are only treated by flotation a combined concentrate is made assaying 55·2 per cent copper, 3·76 ozs. gold, 19·7 ozs. silver, with recoveries of 83·5 per cent of the copper, 80·6 per cent of the gold, and 81·8 per cent of the silver.

If the tailing products from table concentration are also reground and floated, the grade of the combined concentrate is lowered and the recoveries increased. A combined concentrate is made assaying 51·5 per cent copper, 3·75 ozs. gold, 19·25 ozs. silver, with recoveries of 87·7 per cent of the copper, 90·5 per cent of the gold, and 90·2 per cent of the silver.

The re-run middling and tailing and the −70 flotation tailing are high in copper, due to the carbonate content in the ore. A determination was made for carbonate content in the flotation tailing from the re-run of the middling product, which assayed 2·05 per cent copper, and was found to contain 1·70 per cent copper as carbonate.

The tables following give the results obtained from each test. A table of screen analyses is also given for tests Nos. 1, 4, and 5.

Test No.	Product	Weight grams	Per cent weight	Per cent total weight	Cu per cent	Cu % assay x % weight	Au-oz.	Au per cent x oz.	Ag-oz.	Ag per cent x oz.	Per cent of values		
											Copper	Gold	Silver
7	*−18 +35*												
	Table conc...	132	6·7	3·07	61·14	187·7	4·84	14·87	22·86	70·13	30·42	34·15	30·96
	Table tailing	1,569	79·6	36·54	0·95	34·7	0·12	4·38	0·54	19·73	5·62	10·01	8·73
		1,701	86·3	39·61		222·4		19·25		89·86	36·04	44·16	39·69
	−35 +70												
	Table conc...	96	8·5	2·23	60·50	134·8	4·90	10·93	23·12	51·39	21·85	25·08	22·71
	Table tailing	872	77·0	20·29	0·77	15·6	0·08	1·62	0·32	6·49	2·53	3·72	2·87
		968	85·5	22·52		150·4		12·55		57·88	24·38	28·80	25·58
	−18 +70		13·7										
	Middling.....	434	14·5	10·11	5·47	55·3	0·54	5·46	2·16	21·84	8·96	12·52	9·66
	−70												
	Flot. conc.....	148	12·4	3·44	45·53	156·7	1·56	5·36	14·08	48·45	25·40	12·30	21·41
	Flot. tailing..	1,045	87·6	24·32	1·32	32·2	0·04	0·97	0·34	8·27	5·22	2·22	3·666
		1,193	100·0	27·76		188·9		6·35		56·72	30·62	14·52	25·07
	Totals.....	4·296		100·00		617·0		43·59		226·30	100·00	100·00	100·00
	Feed.....	4,300	.....	.....	5·92	592·0	0·46	46·00	2·17	217·00	.....	.....	.....
	Re-run midd.												
	Flot. conc.....	23	5·9	0·60	60·44	36·2	6·57	3·94	23·50	14·10	5·82	9·02	6·70
	Flot. tailing..	367	94·1	9·51	2·05	19·5	0·16	1·52	0·65	6·18	3·14	3·50	2·94
		390	100·0	10·11		55·7		5·46		20·28	8·96	12·52	9·64
	Re-run table tailing.												
	Flot. conc.....	21	2·1	1·19	22·82	27·1	3·67	4·37	15·75	18·77	4·23	9·93	8·40
	Flot. tailing..	978	97·9	55·64	0·45	25·0	0·03	1·67	0·14	7·79	3·92	3·80	3·20
		999	100·0	56·83		52·1		6·04		26·56	8·15	13·73	11·60

Screen Tests

Test No. 1—Screen Analysis of Flotation Tailing

Mesh	Weight grams	Per cent	Cum. per cent
+ 48	13	2·6	2·6
+ 65	17	3·4	6·0
+100	114	22·8	28·8
+150	93	18·6	47·4
+200	98	19·6	67·0
−200	165	33·0	100·0

Test No. 4—Screen Analysis of Table Tailing

Mesh	Weight grams	Per cent	Cum. per cent
+ 48	30	6·1	6·1
+ 65	77	15·7	21·8
+100	109	22·2	44·0
+150	63	12·9	56·9
+200	73	14·9	71·8
−200	138	28·2	100·0

Test No. 5—Screen Analysis of Flotation Tailing

Mesh	Weight grams	Per cent	Cum. per cent
+ 48	38	5·7	5·7
+ 65	118	17·6	23·3
+100	152	22·6	45·9
+150	82	12·2	58·1
+200	103	15·3	63·4
−200	179	26·6	100·0

Conclusions From Results of Experimental Tests

The ore as submitted to the Testing Laboratories is amenable to treatment.

The presence of free gold would indicate that amalgamation should be the first step for the recovery of the values. While the free gold can be recovered in the table concentrate, its presence in the concentrate makes sampling difficult. It is always practicable to recover it by amalgamation as the cost is not excessive, and it provides a quick return of a portion of the values in the ore. Test No. 3 shows that 30 per cent of the gold values can be recovered in this manner.

The copper values are present chiefly as chalcocite with small amounts of the carbonate malachite.

By table concentration and flotation the greater part of the carbonate content is lost in the slime tailing, but the chalcocite content is recovered in a high grade concentrate, a most desirable product for the smelters.

The silver values seem to be closely associated with the copper, and are recovered with the copper in the concentrate. The recoveries of copper and silver are practically identical in the concentrate.

The sizing of the pulp before table concentration results in a higher grade copper concentrate and is worthy of consideration, from a shipping standpoint, in the disposal of the concentrate to the smelters.

The results of the tests show that it is possible to recover 90 per cent of each of the copper, gold, and silver values in the ore. A shipping concentrate assaying 50 per cent to 55 per cent copper is also obtained.

The following flow sheet is suggested for the treatment of the ore:—

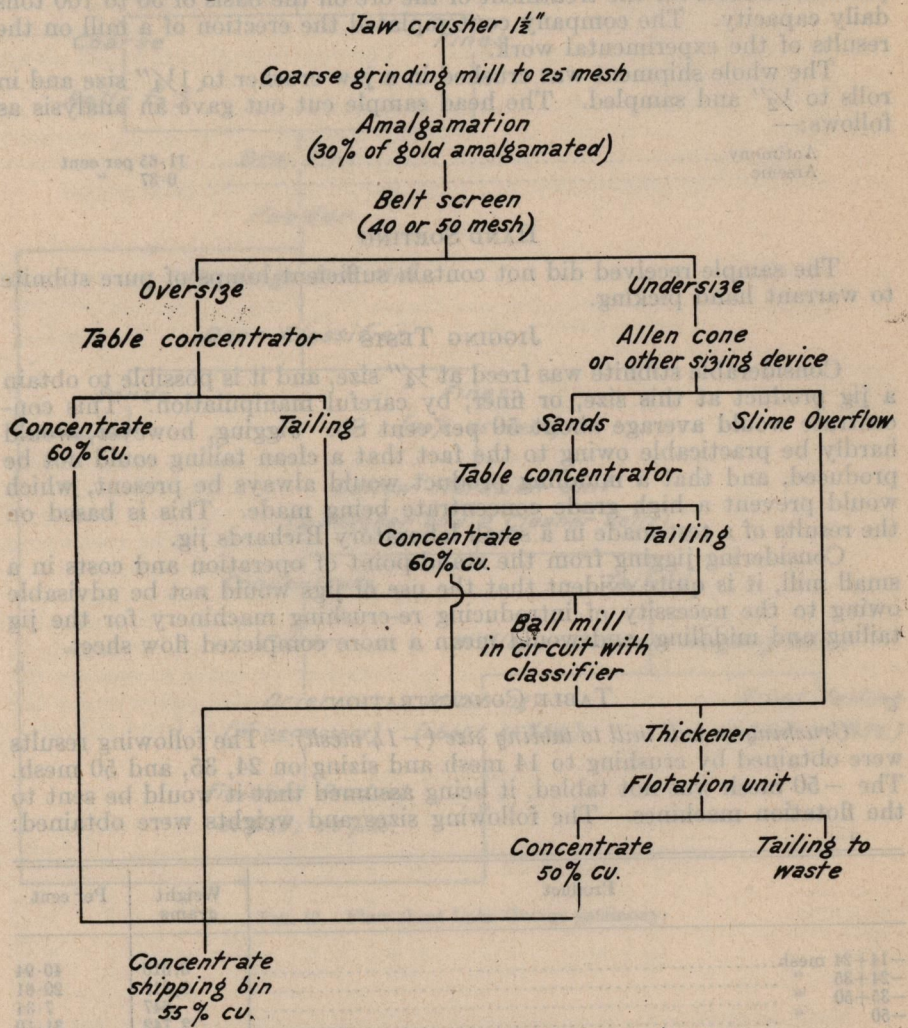


FIG. 11. Flow sheet suggested for treatment of copper-gold ore from Usk, B.C.

Test No. 168

THE CONCENTRATION OF THE ANTIMONY ORE FROM LAKE GEORGE, N.B.

C. S. Parsons

A shipment of 2,154 pounds of antimony ore was received in August 1922 from the North America Antimony Smelting Company, Limited, Lake George, N.B. This shipment was supposed to be a true and representative sample of the milling ore from the company's mine.

Experimental test work was desired to determine a concentration process, suitable for the treatment of the ore on the basis of 50 to 100 tons daily capacity. The company contemplated the erection of a mill on the results of the experimental work.

The whole shipment was crushed in a jaw crusher to 1¼" size and in rolls to ½" and sampled. The head sample cut out gave an analysis as follows:—

Antimony	11·65 per cent
Arsenic	0·37 "

Hand Sorting

The sample received did not contain sufficient lumps of pure stibnite to warrant hand picking.

Jigging Tests

Considerable stibnite was freed at ¼" size, and it is possible to obtain a jig product at this size, or finer, by careful manipulation. This concentrate would average about 50 per cent Sb. Jigging, however, would hardly be practicable owing to the fact that a clean tailing could not be produced, and that a middling product would always be present, which would prevent a high grade concentrate being made. This is based on the results of a test made in a small laboratory Richards jig.

Considering jigging from the stand-point of operation and costs in a small mill, it is quite evident that the use of jigs would not be advisable owing to the necessity of introducing re-crushing machinery for the jig tailing and middling, and would mean a more complexed flow sheet.

Table Concentration

Crushing in ball mill to tabling size (−14 mesh).—The following results were obtained by crushing to 14 mesh and sizing on 24, 35, and 50 mesh. The −50 mesh was not tabled, it being assumed that it would be sent to the flotation machines. The following sizes and weights were obtained:

Product	Weight grams	Per cent
−14+24 mesh	3,610	40·94
−24+35 "	1,817	20·61
−35+50 "	647	7·34
−50 "	2,742	31·10

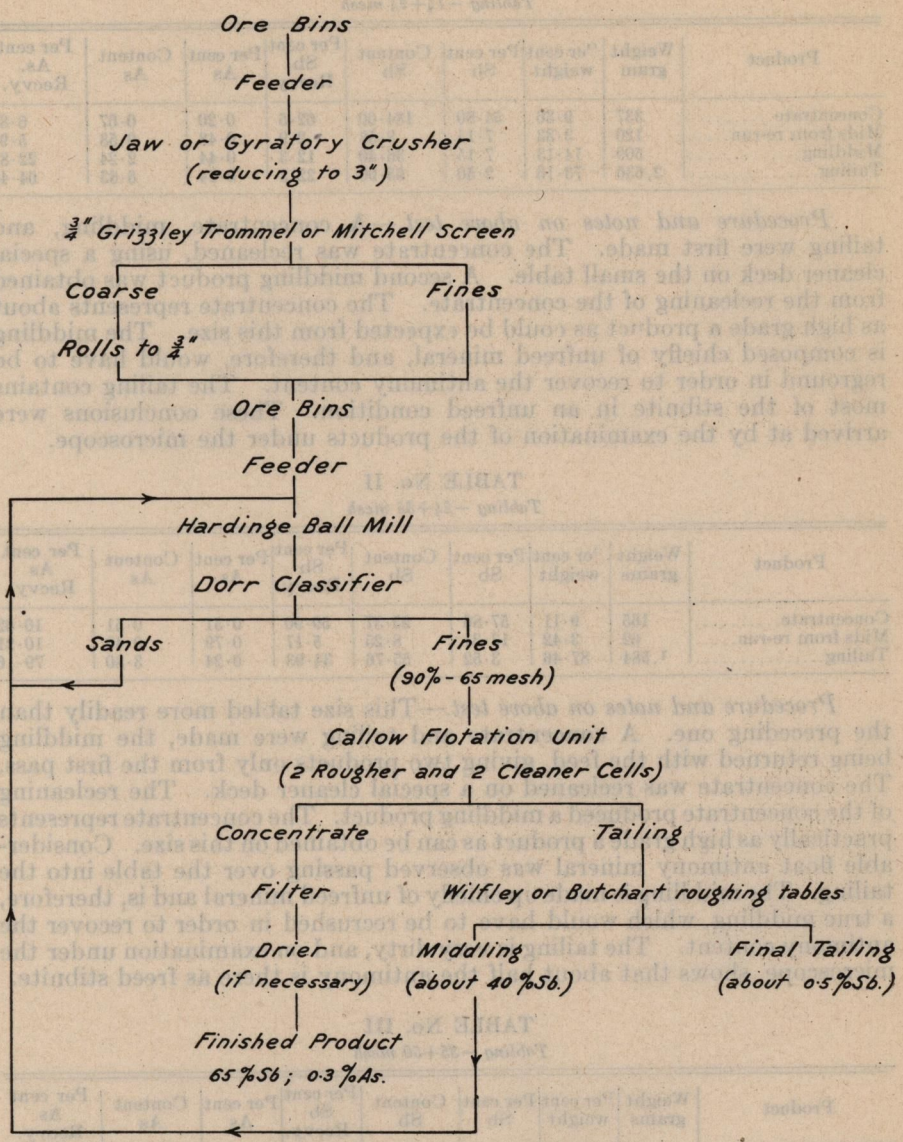


FIG. 12. Flow sheet Lake George antimony.

TABLE No. I

Tabling −14+24 mesh

Product	Weight gram	Per cent weight	Per cent Sb	Content Sb	Per cent Sb Recvy.	Per cent As	Content As	Per cent As. Recvy.
Concentrate	337	9·36	54·80	184·60	62·5	0·20	0·67	6·82
Mids from re-run	120	3·33	7·15	8·58	2·9	0·48	0·58	5·91
Middling	509	14·13	7·15	36·40	12·3	0·44	2·24	22·81
Tailing	2,636	73·16	2·50	65·90	22·3	0·24	6·33	64·46

Procedure and notes on above test.—A concentrate, middling, and tailing were first made. The concentrate was recleaned, using a special cleaner deck on the small table. A second middling product was obtained from the recleaning of the concentrate. The concentrate represents about as high grade a product as could be expected from this size. The middling is composed chiefly of unfreed mineral, and therefore, would have to be reground in order to recover the antimony content. The tailing contains most of the stibnite in an unfreed condition. These conclusions were arrived at by the examination of the products under the microscope.

TABLE No. II

Tabling −24+35 mesh

Product	Weight grams	Per cent weight	Per cent Sb	Content Sb	Per cent Sb Recvy.	Per cent As	Content As	Per cent As Recvy.
Concentrate	165	9·11	57·86	95·37	59·90	0·31	0·51	10·62
Mids from re-run	62	3·42	13·31	8·25	5·17	0·79	0·49	10·21
Tailing	1,584	87·46	3·52	55·76	34·93	0·24	3·80	79·16

Procedure and notes on above test.—This size tabled more readily than the preceding one. A concentrate and tailing were made, the middling being returned with the feed, giving two products only from the first pass. The concentrate was recleaned on a special cleaner deck. The recleaning of the concentrate produced a middling product. The concentrate represents practically as high grade a product as can be obtained on this size. Considerable float antimony mineral was observed passing over the table into the tailing. The middling is made up chiefly of unfreed mineral and is, therefore, a true middling, which would have to be recrushed in order to recover the antimony content. The tailing is very dirty, and on examination under the microscope, shows that about half the antimony is there as freed stibnite.

TABLE No. III

Tabling −35+50 mesh

Product	Weight grams	Per cent weight	Per cent Sb	Content Sb	Per cent Sb Recvy.	Per cent As	Content As	Per cent As Recvy.
Concentrate	63	9·53	58·02	36·55	57·14	0·28	0·18	8·41
Mids from re-run	36	5·45	10·45	3·76	5·88	0·53	0·19	8·88
Tailing	545	82·45	3·74	20·38	31·86	0·31	1·69	78·97
Slimes	17	2·57	19·25	3·27	5·11	0·47	0·08	3·74

Notes on above test.—A concentrate and tailing were made from the first pass of the ore, and the concentrate was recleaned on a special cleaner deck.

CONCLUSIONS FROM TABLE TESTS

It is possible to obtain a table concentrate with an analysis of 55 per cent Sb. if careful sizing is resorted to after crushing to 14 mesh, with a recovery of 55 per cent to 60 per cent of the stibnite in the ore. Here again the middling and tailing products would require regrinding to recover the unfreed antimony. Half the loss in the tailing from the −24 +35 and −35 +50 mesh products was due to flotation of the antimony when tabling. The finer the ore is crushed for tabling, the greater will be the loss from the antimony mineral floating off on the surface of the water, and reporting in the tailing.

Laboratory Flotation Tests on Crude Ore

See Table No. IV

Part of the original head sample was used for these tests. The procedure adopted was to use 1,000 grams of ore crushed dry to 20 mesh and then to approximately 65 mesh in a small ball mill wet.

SCREEN TEST OF FLOTATION FEED

Screen	Weight	Weight per cent	Weight Acc. per cent
+ 65	50	5·2	5·2
− 65+100	204	21·1	26·3
−100+150	137	14·1	40·4
−150+200	168	17·4	57·8
−200	410	42·2	100·0

Test No. 1.—1,000 grams of ore; mixture of 40 per cent coal tar and 60 per cent coal tar creosote, and pine oil for frothing; neutral pulp.

Test No. 2.—1,000 grams of ore; No. 1 K.K. oil from Southwestern Engineering Co., and pine oil for frothing; neutral pulp.

Test No. 3.—1,000 grams of ore; No. 1 K.K. oil and pine oil; pulp alkaline with 4 pounds per ton of soda ash.

Test No. 4.—1,000 grams of ore; No. 1 K.K. oil and pine oil; pulp alkaline with 4 pounds per ton of lime. Note, no stibnite would float in presence of lime.

Test No. 5.—1,000 grams of ore; No. 1 K.K. oil and pine oil; pulp acid with 10 pounds per ton sulphuric acid. Note, this test gave better results.

TABLE No. IV

Test No.	Product	Weight per cent	Analysis		Per cent recovery		Reagents
			Per cent Sb	Per cent As	Sb	As	
1	Concentrate.....	16·0	64·24	0·28	86·7	.....	Coal tar mixture
	Middling.....	2·8	6·38	1·62	1·4	.....	Pine oil
	Tailing.....	81·2	1·75	0·35	11·9	.....	
2	Concentrate.....	17·7	61·80	0·62	81·4	35·6	K.K. oil
	Middling.....	4·0	4·50	1·04	13·4	13·5	Pine oil
	Tailing.....	78·3	0·88	0·27	5·2	50·9	
3	Concentrate.....	15·3	62·70	1·00	83·6	40·0	Soda ash
	Middling.....	4·1	13·10	1·44	4·7	15·4	K.K. oil
	Tailing.....	80·6	1·65	0·21	11·7	44·5	Pine oil
4	No flotation....	.....	.....	.....	.....	.....	Lime, K.K. oil, Pine oil
5	Concentrate.....	17·4	63·36	0·31	95·6	16·4	K.K. oil
	Middling.....	4·3	5·94	1·29	2·2	16·7	H_2SO_4
	Tailing.....	78·3	0·33	0·28	2·2	66·9	Pine oil

CONCLUSIONS

Flotation tests on the crude ore show that by crushing to about 65 mesh, a high grade concentrate 60 per cent to 65 per cent Sb. can be obtained with a recovery up to 95 per cent of the antimony values in the ore. The arsenic content in this concentrate is about 0·3 per cent. If no objectionable feature arises in the handling of this fine concentrate in the subsequent treatment, this would seem to be the simplest and most feasible method of treating the ore. An outline of such a flow sheet would be as follows:

Coarse crushing in breaker and rolls to $\frac{3}{4}''$ to $\frac{1}{2}''$, and ball mill in closed circuit with classifier of Dorr type, to approximately 65 mesh. The overflow of the classifier would go to flotation machines, say of the Callow type, making a rougher concentrate which would be recleaned on a series of cleaner cells, and a tailing, which would go to roughing tables. These roughing tables are recommended on account of the high grade feed going to cells. They would act as a guard on the operation of the cells. The concentrate from the tables would be fed back to the circuit at the ball mill.

These deductions are based on the results obtained from the sample of ore received, and do not take into consideration any local conditions at the mine, or special market conditions regarding the physical character of the concentrate required for subsequent treatment. The process which gave the highest grade concentrate, the highest recovery, with simplicity of operation is recommended

In the construction of a small concentrator of 100 tons daily capacity, simplicity of design is important. Jigging and tabling followed by flotation would mean extra regrinding, classification, sizing, and thickening

units, giving a lower grade final product, but a product much coarser than that obtained by straight flotation, and in this respect may be more suitable for subsequent refining. The final recovery of antimony values would be about the same in both cases. (See flow sheet, page 137.)

Large Scale or Tonnage Check Tests

A large scale, or tonnage check test, was made in a small pilot Callow flotation unit consisting of two rougher cells of the new flat bottom type, and two cleaner cells of the same design. The ore was reduced to ½" in a jaw crusher and rolls, and fed to a Hardinge conical ball mill in closed circuit with a Dorr classifier. The classifier was adjusted to give a −50 mesh product to the flotation cells. The test was made on 1,800 pounds of ore.

The concentrate assayed	58·21 per cent antimony
The tailing assayed	0·94 " "

The recovery is figured from the formula $R = \frac{(H - t)\,C}{(C - T)\,H} \times 100 = 93 \cdot 5\%$.

That is, 93·5 per cent of the antimony was recovered in a concentrate assaying 58·2 per cent antimony.

H = Assay of feed
T = Assay of tailing
C = Assay of concentrate

The reagents used for flotation were a mixture of 20 per cent coal tar and 80 per cent coal tar creosote, from the Dominion Tar and Chemical Co., Sault Ste. Marie, Ont., and sufficient steam distilled pine oil to maintain a good froth. Five pounds of sulphuric acid was added per ton of ore just before entering the cell.

Conclusions

The results from this test confirm the results obtained from the small scale laboratory tests. The grade of the concentrate is slightly lower, but this does not mean that a concentrate of higher grade, similar to that obtained in the small tests, cannot be obtained. After the operator becomes familiar with the operation of the flotation unit, better work can be expected. It is difficult to gauge the grade of the concentrate being produced, but with training and efficient manipulation, similar results to those of the small tests should easily be obtained.

Test No. 169

THE RECOVERY OF THE SILVER VALUES IN THE LEACHED CHLORIDIZED RESIDUES OF THE DOMINION REDUCTION COMPANY, COBALT, ONT.

C. S. Parsons and C. L. Dewar

A shipment of 15 bags of silver residues was received July 25, 1922, at the Ore Dressing and Metallurgical Laboratories, from the Dominion Reduction Company, Cobalt, Ont.

The material is the residue from the chloridizing, roasting and leaching of flotation and table concentrates produced by Cobalt mills, and contains silver to the extent of 20 to 30 ozs. per ton. It was desired that tests be conducted to determine, if possible, some satisfactory process for the recovery of the silver.

Head Samples

Head samples from bags 1 and 2 were analysed with the following results:—

—	Bag 1	Bag 2
Silver	Oz./ton 29·0	Oz./ton 21·68
Copper	per cent	per cent
Sulphur	0·75	0·77
Arsenic	1·36	0·73
Cobalt	0·82	0·94
Silica	0·40	0·60
Alumina	41·70	41·45
Lime	.....	23·43
Magnesia	.....	2.60
Carbon	.....	3·42
Chlorides	.....	0·47
	trace	trace

The following approximately complete analysis has been calculated upon the assumption that the metals are present as oxides:—

	Bag 2
Fe_2O_3	17·16 per cent
Al_2O_3	23·43 "
SiO_2	41·45 "
CaO	2·60 "
MgO	3·42 "
SO_3	1·83 "
As_2O_3	1·24 "
Co_3O_4	0·78 "
CuO	0·96 "
C	0·47 "
Ag	0·07 "
Total	93·41 "

(Alkalies not determined)

A screen test on a sample from one bag of residues gave the following results:—

Mesh	Weight grams	Weight per cent	Weight cum. per cent
3	12	2·1	2·1
6	92	16·3	18·4
10	122	21·6	40·0
20	73	12·9	52·9
35	56	9·9	62·8
65	50	8·9	71·7
150	44	7·8	79·5
−150	116	20·5	100·0

Test 1.—This test was made to determine if any separation could be effected by classification. A grab sample was taken from bag 1, and screened on 10 mesh, as the +10 mesh material would be too coarse for the small Richards pulsator classifier used.

Weight of sample: 2500 grams Moisture 16 p.c.
Dry weight: 2100 " Silver 33·0 oz./ton.

Product	Weight grams	Weight per cent	Silver oz./ton	Per cent of total
+10 mesh	570	27·2	18·43	15·1
Class 1	375	17·9	21·14	11·4
" 2	177	8·4	23·16	5·9
Overflow	665	31·7	43·35	41·5
Slime	114	5·4	53·10	8·7
Loss (calculated)	199	9·4	60·8	17·3

The loss was very fine slime that would not settle in the slime box. The test shows that no separation could be made by classification.

Flotation

Test 2.—A sample of 1,175 grams (15 per cent moisture) was taken and ground in a ball mill for half an hour with 0·7 lb. per ton of a coal tar mixture (coal tar 40 per cent, coal tar creosote 60 per cent) and 0·2 lb. per ton of Barrett No. 4, then floated in a Ruth machine. In this test there was too much coarse material, so for subsequent tests the residues were crushed to −14 mesh and then ground half an hour in the ball mill. In all nine flotation tests were made. The conditions of tests and data desired therefrom are given in tabular form in the accompanying table. (Page 146.) As a whole the results were not very satisfactory, for in all tests the recovery in concentrate was less than 66 per cent, and in the majority it did not exceed 50 per cent. It is possible that the recovery might be increased by the retreatment of the middling, but in any case the tailing would still carry 11·0 or more ozs. per ton.

From the results obtained it is deemed impossible to satisfactorily treat these residues by flotation alone.

Amalgamation

Test 14.—A 1,000 gram sample of −35 mesh residues was ground in the pebble mill with 100 grams of mercury and 5 grams of salt (NaCl) for 2 hours. The pulp was panned and the mercury recovered.

Silver amalgamated..... 13·4 per cent

Russell Process

Test 8.—A 200 gram sample of −14 mesh residues was ground for half an hour in the ball mill, then agitated with 1,000 c.c. of solution (0·75 per cent $CuSO_4$ and 1·25 per cent $Na_2S_2O_3$) for 18 hours.

Feed..... Ag 29·0 oz./ton
Tailing..... Ag 23·9 " ..... Extraction 17·6 per cent

Cyanidation Tests

Test 13.—A 200 gram sample of tailing from flotation test 7 was agitated with 100 c.c. water, 5 grams lime, and 5 grams cyanide (25 per cent NaCN) for 26 hours.

Feed	Ag 11·7 oz./ton		
Tailing	Ag 8·26 "	Extraction	29·4 per cent
Recovery by flotation (conc.—mids.)			69·2 "
Recovery by cyanidation			9·1 "
Total recovery			78·3 "

Test 15.—A 500 gram sample of −200 mesh residues from bag 2 was agitated with 3,000 c.c. of 3 per cent sulphuric acid solution for 24 hours. It was then washed, made alkaline, cyanide added and agitated for 48 hours.

Solution at beginning of agitation	0·40 per cent CaO, 0·14 per cent KCN		
Feed	29·0 oz./ton Ag		
Tailing	5·96 oz./ton Ag	Extraction	79 per cent

Test 17.—A 750 gram sample of −200 mesh residues was agitated with 2,200 c.c. of sulphuric acid solution (3 per cent H_2SO_4) for 5 hours, washed, made alkaline, cyanide added and agitated for 27 hours. The agitation was for 8 hours each day, and the total time of contact was 80 hours.

Solution at beginning of agitation	0·01 per cent CaO, 0·05 per cent KCN		
Feed	29·0 oz./ton Ag		
Tailing	9·5 oz./ton Ag	Extraction	67 per cent

Test 19.—A 500 gram sample of −200 mesh residues was agitated with sulphuric acid solution (2,250 c.c. of 6 per cent H_2SO_4) for 6 hours, washed, made alkaline, cyanide added and agitated for 85 hours.

Feed	21·68 oz./ton Ag		
Tailing	5·72 oz./ton Ag	Extraction	73·6 per cent

Test 20.—A 500 gram sample of −200 mesh residues was agitated in cyanide solution for 85 hours.

Feed	21·68 oz./ton Ag		
Tailing	6·94 oz./ton Ag	Extraction	68 per cent

Test 21.—A 500 gram sample of −200 mesh residues was agitated with 4,000 c.c. solution containing 20 grams bleaching powder (commercial) washed and decanted, then agitated with cyanide solution for 85 hours:—

Feed	21·68 oz./ton Ag		
Tailing	6·44 oz./ton Ag	Extraction	70 per cent

Test 22.—It was thought that the coal in the residues was precipitating part of the silver dissolved. To determine whether this was the case or not, the coal was floated with pine oil No. 5 from 500 grams residues.

The cleaned residues were then agitated with sulphuric acid solution (3½ per cent H_2SO_4) for 40 hours, then cyanided for 95 hours:—

Feed.....	21·68 oz./ton Ag		
Tailing.....	6·98 oz./ton Ag.....	Extraction	68 per cent

1,000 c.c. of the pregnant solution was divided into two parts A and B. 400 c.c. of A was put through a filter containing very finely ground coal. The filtrate was evaporated and the residue assayed. The B part was merely assayed.

	A	B
Silver gm/litre	66·0	74·0

These results showed that the effect of the small amount of coal present in the residues is negligible.

SUMMARY AND CONCLUSIONS

(1) No separation was effected by classification.

(2) Flotation alone does not give very satisfactory results.

(3) Amalgamation gives the very low recovery of 13 per cent.

(4) Hyposulphite lixiviation was not successful in recovering the silver.

(5) Cyanidation alone gives a fair recovery—68 per cent—which can be increased by a preliminary acid wash (70-79 per cent).

(6) The recovery by flotation and cyanidation of the flotation tailing was the same as that obtained by acid washing and cyaniding for about twice the length of time. It should be noted, though, that the flotation concentrate would require further treatment while the silver dissolved by the cyanide is readily recovered as precipitate. This is a point in favour of all cyanidation.

(7) Bleaching powder is not so efficacious as the acid for a preliminary wash, but it requires much less lime or soda ash to produce the required protective alkalinity for subsequent cyanidation.

(8) The small amount of coal in the residues has little effect upon the recovery of the silver by cyanidation.

This concludes the test work completed in 1922, but as the recoveries obtained were not sufficiently high, other tests will be made.

Flotation Tests

Silver residues. Dominion Reduction Co.

Test No.	Product	Weight per cent	Silver oz./ton	Silver per cent x oz.	Recoveries	Remarks
2	Concentrate...	18·9	105·1	1,985	65·9	Coal tar and coal tar creosote (40 per cent and 60 per cent) 0·7 lb./ton. Barrett No. 4, 0·2 lb./ton.
	Tailing.....	81·1	12·8	1,053	34·1	
3	Concentrate...	4·8	76·8	383	12·6	−14 mesh material ground for half hour with coal tar mixture 0·7 lb./ton. Barrett No. 4, 0·2 lb./ton. Floated with Na_2S, $9H_2O$, 8 lb./ton and P.T.T. No. 350, 0·1 lb./ton.
	Middling.....	19·5	46·6	937	30·7	
	Tailing.....	75·7	21·6	1,728	56·7	
4	Concentrate...	5·5	145·5	800	27·5	Coal tar mixture 0·7 lb./ton. Barrett No. 4, 0·25 lb./ton. Caustic soda 4·0 lb./ton. P.T.T. No. 350, 0·2 lb./ton.
	Middling.....	14·0	36·6	512	17·6	
	Tailing.....	80·5	19·8	1,594	54·9	
5	Concentrate...	4·3	81·5	351	12·3	Ground 25 min. with coal tar mixture 0·7 lb./ton. Barrett No. 4, 0·25 lb./ton. Then 5 min. with 8·0 lb./ton Na_2S $9H_2O$. Floated with P.T.T No. 350, 0·1 lb./ton. Concentrate cleaned with Barrett No. 4 and W.G. tar.
	Middling.....	14·9	42·4	632	22·2	
	Tailing.....	80·8	23·2	1,875	65·5	
7	Concentrate...	5·1	287·0	1,463	49·4	Coal tar mixture 1·8 lb./ton. Pine oil No. 5, 0·15 lb./ton. Concentrate cleaned with Barrett No. 4, 0·15 lb./ton and Pine oil No. 4.
	Middling.....	16·8	35·0	588	19·8	
	Tailing.....	78·1	11·7	914	30·8	
9	Concentrate 1.	21·0	78·4	1,646	57·3	Ground half hour with coal tar mixture 1·15 lb./ton, No. 28 F.P.L. Ketone to froth, 4 lb./ton caustic soda. Made first concentrate then added water and 4 lb./ton $CuSO_4$ and little Ketone.
	Concentrate 2.	10·3	40·6	419	14·6	
	Tailing.....	68·7	11·7	804	28·1	
10	Concentrate...	16·9	50·0	845	30·8	Coal tar mixture 1·15 lb./ton and NaCl 4·0 lb./ton (both to mill). Pine oil No. 5, 0·15 lb./ton to cell.
	Tailing.....	83·1	22·8	1,899	69·2	
11	Concentrate...	18·0	49·0	882	33·7	Coal tar mixture 1·15 lb./ton and NaCl 4·0 lb./ton (both to mill). 10 c.c. 40 per cent solution Na_2S $9H_2O$ (agitated in cell 7 minutes). Pine oil No. 5, 0·15 lb./ton. Pulp neutral to litmus.
	Tailing.....	82·0	21·2	1,738	66·3	
12	Concentrate...	17·9	74·8	1,339	51·6	Coal tar mixture. Pine oil No. 5. NaCl 4·0 lb./ton. 10 c.c. 40 per cent solution of Na_2S $9H_2O$.
	Tailing.....	82·1	15·3	1,256	48·4	

Test No. 170

GOLD ORE FROM BEAR RIVER, BEDWELL SOUND, VANCOUVER I.

R. K. Carnochan

A shipment of 198·5 pounds of gold ore was received at the Ore Dressing and Metallurgical Laboratories on September 14, 1922, from J. B. Woodworth, Vancouver, B.C. The ore was obtained from a mining property near Bear river, Bedwell sound, Vancouver island, and consisted of mineralized quartz. Sphalerite, galena, pyrite, chalcopyrite, and arsenopyrite are present in varying amounts. Silver and gold are present, and while these are closely associated with the metallic sulphides, free gold is noticeable on close examination.

The presence of metallic gold in the ore makes sampling somewhat difficult. An average of four samples gave the following analysis:—

Gold	Au	5·63 ounces.
Silver	Ag	4·28 "
Arsenic	As	0·41 per cent.
Copper	Cu	0·42 "
Lead	Pb	0·78 "
Zinc	Zn	4·70 "
Iron	Fe	9·33 "
Sulphur	S	10·00 "
Silica	SiO_2	69·00 "
Alumina	Al_2O_3	2·26 "
Lime and magnesia	CaO, MgO	trace

Experimental test work was desired to determine a suitable method of treatment, and as the chief value is in its gold content, the recovery of the gold is of primary importance and the other values secondary.

The test work conducted on the ore was as follows:—

First: Amalgamation
Second: Cyanidation.
Third: Table concentration and cyanidation of the table tailing.
Fourth: Table concentration, flotation, and cyanidation of the tailing.

Amalgamation

Test No. 1.—1,000 grams of heads −40 mesh were ground in a small pebble jar for 3 hours with 100 grams of mercury and 400 c.c. of water. The charge was then panned and 83·7 grams of mercury was recovered. The tailing was dried and sampled for assay.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	2·70	3·34	48·0	78·0
Amalgamated	2·93	0·94	52·0	22·0
Heads −40	5·63	4·28	100·0	100·0

Test No. 2.—This test was made similar to test No. 1, with 5 grams of sodium hydroxide added to the mill charge. The mercury recovered weighed 92·2 grams.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	1·96	3·04	34·8	71·0
Amalgamated	3·67	1·24	65·2	29·0
Heads −40	5·63	4·28	100·0	100·0

Test No. 3.—500 grams of heads, out of which some metallics had been removed, were reduced to −200 mesh and then ground for 3 hours in a small pebble jar with 50 grams of mercury, 200 c.c. of water, and 10 grams of sodium hydroxide. The charge was then panned and 47·9 grams of mercury were recovered. The tailing was dried and sampled for assay.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	1·92	3·06	34·1	71·5
Metallics	0·17	0·04	3·0	0·9
Amalgamated	3·54	1·18	62·9	27·6
Heads −200	5·63	4·28	100·0	100·0

Total recovery Au ..... 65·9 per cent.
Total recovery Ag ..... 28·5 "

SUMMARY AND CONCLUSIONS FROM RESULTS OF AMALGAMATION TESTS

The above results show that 50 per cent to 65 per cent of the gold values can be recovered by amalgamation, crushing the ore to 40 mesh. Finer grinding as in test No. 3 does not seem to improve the results. The consumption of mercury was high in all these tests, and would be prohibitive in practice, but this may be due to some foreign matter, such as oil, getting into the sample, as the addition of sodium hydroxide reduces the mercury loss and improves the recovery. Amalgamation for the recovery of 50 per cent of the gold values in the ore may, therefore, be feasible, as it entails very little additional cost of installation, and provides an early return of one-half of the values in the ore.

CYANIDATION

Test No. 4.—1,142 grams of the ore −100 mesh was agitated for 82 hours in 6,000 c.c. of water to which 15 grams of sodium cyanide and 10 grams of lime had been added.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	1·36	2·53	24·2	59·1
Cyanided	4·27	1·75	75·8	40·9
Heads −100	5·63	4·28	100·0	100·0

SUMMARY AND CONCLUSIONS FROM RESULTS OF CYANIDATION TEST

Seventy-five per cent of the gold and 40 per cent of the silver was recovered by straight cyanidation of the ore after grinding to 100 mesh. The consumption of cyanide was high, and the recoveries would be considered low on such a high grade ore. Grinding to 200 mesh would improve the extraction. As a large proportion of the precious metals values is very closely associated with the sulphides in the ore, a long time of agitation is necessary for the extraction of these values.

TABLE CONCENTRATION AND CYANIDATION OF THE TABLE TAILING

Test No. 5.—1,252 grams of heads −40 mesh were tabled on a small Wilfley table, making a concentrate and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Table concentrate	259	22·34	15·57	5,786	4,033	82·1	75·3
Table tailing	975	0·70	1·32	682	1,287	9·7	24·0
Loss	18	32·28	2·17	581	39	8·2	0·7
Heads −40	1,252	5·63	4·28	7,049	5,359	100·0	100·0

Eight hundred and fifty grams of the table tailing were ground to pass 100 mesh and agitated for 8 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tails	0·42	0·66	5·8	12·0
Cyanided	0·28	0·66	3·9	12·0
Table tail −100	0·70	1·32	9·7	24·0

Product	Per cent Au value	Per cent Ag value
Table concentrate	82·1	75·3
Cyanided	3·9	12·0
Total recovery	86·0	87·3

Test No. 6.—1,307 grams of heads −80 mesh were tabled on a small Wilfley table making a concentrate and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Table concentrate	302	20·60	13·39	6,221	4,044	84·5	72·3
Table tailing	922	0·66	1·20	608	1,106	8·3	19·8
Loss	83	6·37	5·35	529	444	7·2	7·9
Heads −80	1,307	5·63	4·28	7,358	5,594	100·0	100·0

Eight hundred and seven grams of the table tailing was ground to pass 100 mesh and agitated for 8 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tailing	0·46	0·84	5·8	13·9
Cyanided	0·20	0·36	2·5	5·9
Table tailing −100	0·66	1·20	8·3	19·8

Product	Per cent Au value	Per cent Ag value
Table concentrate	84·5	72·3
Cyanided	2·5	5·9
Total recovery	87·0	78·2

SUMMARY AND CONCLUSIONS FROM RESULTS OF TABLE CONCENTRATION AND CYANIDATION OF THE TABLE TAILING

By table concentration 80 per cent to 85 per cent of the gold values and 70 per cent to 75 per cent of the silver values are recovered in a high grade concentrate, assaying over 20 ozs. in gold and about 15 ozs. in silver. This is a marketable concentrate that could be disposed of to the smelter. By cyanidation of the table tailing after grinding to 100 mesh, 2·5 per cent to 4 per cent additional gold values are recovered. Finer grinding to 200 mesh for cyanidation would improve the extraction, but these results show the difficulty of obtaining good extractions by cyanidation, due to the close association of the precious metal values with the sulphides in the ore.

TABLE CONCENTRATION, FLOTATION AND CYANIDATION OF THE TAILING

Test No. 7.—2,322 grams of heads −40 mesh out of which some metallics had been removed, were tabled on a small Wilfley table, making a concentrate and a tailing. The table tailing was then floated in a small Ruth machine, making a concentrate, a middling, and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Metallics	.....	·17	·04	295	93	3·0	0·9
Table conc.	562	20·20	15·68	11,352	8,812	86·3	82·7
Flot. conc.	172	7·94	9·94	1,366	1,710	10·4	16·0
Flot. midd.	103	·46	·44	47	45	0·3	0·4
Flot. tail	1,452	trace	trace	.....	.....	.....	.....
Loss	33	.....	.....	.....	.....	.....	.....
	.....	.....	.....	13,160	10,660	.....	.....
Heads −40	2,322	5·63	4·28	13,073	9,938	100·0	100·0

1,303 grams of the flotation tailing was ground to pass 200 mesh and agitated for 15 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime. The tailing from this operation gave only a faint trace of gold and silver upon assay.

Product	Per cent Au value	Per cent Ag value
Metallics	3·0	0·9
Table concentrate	86·3	82·7
Flotation concentrate	10·4	16·0
Total recovery	99·7	99·6

The flotation middling should have been put with the flotation tailing and both cyanided together. This would have increased the recovery.

Test No. 8.—2,292 grams of heads −80 mesh, out of which some metallics had been removed, were tabled on a small Wilfley table, making a concentrate and a tailing. The table tailing was then floated in a small Ruth machine, making a concentrate, a middling, and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Metallics	.....	0·17	0·04	390	92	3·3	0·9
Table conc.	571	18·22	14·86	10,404	8,485	86·9	84·3
Flot. conc.	248	4·52	5·78	1,121	1,433	9·4	14·2
Flot. midd.	114	0·26	0·32	30	36	0·2	0·4
Flot. tail	1,175	0·02	0·02	23	23	0·2	0·2
Loss	184	.....	.....	.....	.....	.....	.....
	.....	.....	.....	11,968	10,069	.....	.....
Heads −80	2,292	5·63	4·28	12,904	9,810	100·0	100·0

One thousand and forty-eight grams of the flotation tailing was ground to pass 200 mesh and agitated for 8 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tailing	trace	trace	.....	.....
Cyanided	0·02	0·02	0·2	0·2
Flotation tailing −200	0·02	0·02	0·2	0·2

Product	Per cent Au value	Per cent Ag value
Metallics	3·3	0·9
Table concentrate	86·9	84·3
Flotation concentrate	9·4	14·2
Cyanided	0·2	0·2
Total recovery	99·8	99·6

The flotation middling should have been put with the flotation tailing, and both cyanided together. This would have given a better recovery.

Test No. 9.—160 pounds of ore, −40 mesh, out of which some metallics had been removed, was tabled on a large Wilfley table, making a concentrate and a tailing. The tailing was then floated in one compartment of a large Ruth machine, making a concentrate and a tailing. The flotation concentrate was then re-run in a small Ruth machine, making a clean concentrate and a middling.

Product	Weight pounds	Au oz./ton	Ag x oz./ton	Au pounds x oz./ton	Ag pounds x oz./ton	Per cent Au value	Per cent Ag value
Metallics	.....	0·17	0·04	27·2	6·4	3·0	0·9
Table conc	40·25	21·16	11·83	851·7	476·2	94·5	69·5
Flot. conc	2·90	3·52	6·02	10·2	17·5	1·1	2·6
Flot. midd	6·10	0·26	0·34	1·6	2·1	0·2	0·3
Flot. tail	95·75	0·08	0·22	7·7	21·1	0·9	3·1
Loss	15·00	0·16	10·77	2·4	161·5	0·3	23·6
Heads −40	160·00	5·63	4·28	900·8	684·8	100·0	100·0

Fifteen hundred grams of flotation middling and flotation tailing mixed in proper proportion were ground to pass 200 mesh and agitated for 22 hours with 6000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tailing	0·00	0·00	0·0	0·0
Cyanided	0·09	0·23	1·1	3·4
Flot. middling and tailing −200	0·09	0·23	1·1	3·4

Product	Per cent Au values	Per cent Ag values
Metallics	3·0	0·9
Table concentrate	94·5	69·5
Flot. concentrate	1·1	2·6
Cyanided	1·1	3·4
Total recovery	99·7	76·4

SUMMARY AND CONCLUSIONS FROM RESULTS OF TABLE CONCENTRATION, FLOTATION, AND CYANIDATION OF THE TAILING

The results of the above three tests show a complete recovery of the precious metal values in the ore. Over 99 per cent of the values are recovered by table concentration and flotation in a high grade concentrate, assaying over 20 ozs. in gold, and about 15 ozs. in silver. By cyanidation of the tailing the remaining precious metals values are extracted. Cyanidation would probably not be necessary, as practically all the values are recovered by concentration. In test No. 9 there is shown a recovery of 69·5 per cent of the silver values in the table concentrate. This does not check with the two former tests, where 82·7 per cent and 84·3 per cent are recovered in similar concentrates, and is probably due to volatilization of some of the silver in making the determination.

Treatment of Concentrate

A complete analysis of the table concentrate obtained from test No 7, was as follows:—

Gold	Au	20·20 ozs.
Silver	Ag	15·68 "
Arsenic	As	1·57 per cent
Copper	Cu	0·92 "
Lead	Pb	2·50 "
Zinc	Zn	15·20 "
Iron	Fe	31·67 "
Sulphur	S	39·28 "
Silica	SiO_2	8·00 "
Alumina	Al_2O_3	0·72 "
Lime and magnesia	MgO, CaO	trace

The above analysis would be typical of the concentrates obtained by table concentration and flotation on the grade of ore submitted for test purposes. If amalgamation was introduced before table concentration and flotation, the gold and silver values in the concentrate would be reduced by the amount of these values recovered by amalgamation. Otherwise the grade would be about the same.

Test No. 10—Cyanidation of concentrates.—1,500 grams of table and flotation concentrate from test No. 9 mixed in proper proportions, were ground to pass 200 mesh and agitated for 174 hours in 6,000 c.c. of water, to which were added 45 grams of sodium cyanide and 40 grams of lime. The tailing from this treatment assayed: gold, 2·00 ozs. per ton; silver, 4·58 ozs. per ton. This tailing was agitated again for 119 hours in 6,000 c.c. of water, to which were added 15 grams of sodium cyanide and 10 grams of lime. The tailing from this second agitation assayed: gold, 0·25 ozs. per ton; silver, 2·48 ozs. per ton. The consumption of cyanide was approximately 31 pounds per ton of concentrate.

Product	Au oz./ton	Ag oz./ton	Per cent Au values	Per cent Ag values
Cyanided	19·72	8·96	98·7	78·3
Tailing	0·25	2·48	1·3	21·7
Concentrate	19·97	11·44	100·0	100·0

A test was conducted by cyaniding the roasted concentrate, but the results were not encouraging.

SUMMARY AND CONCLUSIONS FROM RESULTS OF TREATMENT OF CONCENTRATES

The gold values in the concentrates can be recovered by cyanidation. Fine grinding to 200 mesh is necessary. Better results are obtained by cyaniding the raw concentrates than by roasting before cyanidation. The results show a recovery of 98·7 per cent of the gold values and 78·3 per cent of the silver values in the concentrates, which is considered a high recovery for this class of material. The time of agitation is excessive and the consumption of cyanide is fairly high, but both these could no doubt be considerably reduced in practice by better agitation and the addition of the proper amounts of reagents.

CONCLUSIONS ARRIVED AT FROM RESULTS OF EXPERIMENTAL TESTS

The ore as submitted to the testing laboratories is amenable to treatment.

By amalgamation after crushing to 40 mesh, 50 per cent of gold values and 25 per cent of the silver can be recovered.

By table concentration and flotation, 99 per cent of the gold values and 99 per cent of the silver values can be recovered in the form of a concentrate of metallic sulphides assaying 20 ozs. gold, 15 ozs. silver.

The concentrate can be disposed of to the smelters without further treatment.

The concentrate can be cyanided with a high recovery of the gold content.

In the treatment of this ore, two methods have to be considered, depending largely on local conditions, such as transportation and freight rates, favourable smelter treatment charges, etc.

First:—The reduction of the ore to 40 mesh, table concentration, and flotation of the table tailing. Whether it is more profitable to dispose of the high grade concentrate thus produced to the smelters, or to grind it to 200 mesh and extract the precious metal values by cyaniding.

Second:—The reduction of the ore to 40 mesh, amalgamation to recover the free gold, table concentration and flotation of the table tailing. The disposal of the high grade concentrate thus produced, which will contain about 10 ozs. of gold after amalgamation of the ore, or to regrind it to 200 mesh, and extract the precious metal values by cyaniding.

Test No. 171

THE CONCENTRATION OF THE RADIO-ACTIVE MINERALS IN A SHIPMENT OF PEGMATITE FROM KEARNEY, ONT.

R. K. Carnochan

A shipment of 251 bags of pegmatite supposed to contain radio-active minerals, gross weight 19,019 pounds, was received on October 9, 1922, at the Ore Dressing and Metallurgical Laboratories, from Messrs. Ryan and Mann, Kearney, Ont.

The pegmatite is composed of coarsely crystallized red feldspar quartz and black mica. No radio-active mineral could be seen in hand specimens of the pegmatite.

Tests were desired to determine if the pegmatite contained sufficient radio-active minerals to make it a commercial source of supply of radium, and to determine if the radio-active minerals could be recovered by concentration with the production of a marketable concentrate.

The shipment as received was divided into two lots marked No. 1 and No. 2. Lot No. 1 consisted of 220 bags, net weight 16,449 pounds, and Lot No. 2 contained 31 bags, net weight 2,372 pounds.

Lot No. 1

The whole lot was crushed by a breaker and rolls to ½" and a 177 pound sample cut out by means of a Vezin sampler. The 177 pounds was crushed to −40 mesh by means of a small breaker and a small set of rolls. In screening on 40 mesh a lot of mica was obtained as oversize. The −40 mesh and the +40 mica products were cut by means of a Jones riffle into the following:—

Head sample—	−40	·672	pound
	+40 mica	·012	"
		·684	"
Test portion—	−40	20·5	pounds
	+40 mica	·375	"
		20·875	"
Remainder—	−40	151·5	pounds
	+40 mica	2·75	"
		154·25	"

The 20·5 pounds of −40 of the test portion was run on a small Wilfley table making a concentrate, middling, and tailing. The middling was re-run 3 times. In the last re-run any middling produced was put with the tailing. This gave only three products from the tabling—a concentrate, a tailing, and a slime product. These products were dried. The table concentrate was treated by a hand magnet to remove any very magnetic material, and then put through an Ullrich magnetic separator, making a concentrate and tailing. The table tailing was put through the Ullrich magnetic separator also making a concentrate and tailing. Samples of the feed and all products of the test work were tested for radio-activity, and the following table shows the results obtained:—

Product	Weight pound	Radio-activity
+40 mica	·375	nil
Magnetic from table concentrate	·009	"
Ullrich conc. from table conc	·030	1·52
Ullrich tailing from table conc	·062	0·46
Ullrich conc. from table tailing	·187	nil
Ullrich tailing from table tailing	17·000	"
Table slime	1·718	"
Loss	1·494	.....
Feed	20·875	nil

In testing the radio-activity of the above products, the standard used was a sample of low grade carnotite ore containing about 2 per cent uranium oxide. This standard has a radio-activity of 6·90.

Lot No. 2

This lot, after the removal of a few specimens, was crushed to ½" and a sample of 240·5 pounds cut out by means of a Vezin sampler. The sample was crushed to −40 mesh and a test portion of 15 pounds was cut by a Jones riffle. This test portion was tabled in the same way as described under lot No. 1. The table concentrate and tailing were each run through the Ullrich magnetic separator. Samples of the feed and all products of the test work were tested for radio-activity. The results are shown in the following table:—

Product	Weight pound	Radio-activity
Ullrich concentrate from table concentrate	0·995	0·56
Ullrich tailing from table concentrate	0·086	0·23
Ullrich concentrate from table tailing	0·123	0·04
Ullrich tailing from table tailing	12·500	0·01
Table slime	1·312	0·08
Loss	0·884	.....
Feed	15·000	0·01

The same standard was used to test the radio-activity of the products of the test work on lot No. 2 as was used in testing the products of the test work on lot No. 1.

CONCLUSIONS

The best product obtained in testing lot No. 1 is less than one-fourth as radio-active as a low grade carnotite ore containing 2 per cent uranium oxide, and represents 0·14 per cent of the sample treated.

The best product obtained in treating lot No. 2 is less than one-twelfth as radio-active as a low grade carnotite ore containing 2 per cent uranium oxide, and represents 0·63 per cent of the sample treated.

From the above it is evident that neither of the lots submitted can be considered as ores of radium.

Test No. 172

THE PRECIOUS METALS AND OTHER VALUES IN THE COPPER-NICKEL ORES OF SHEBANDOWAN LAKE, ONT.

In the twenty-ninth Annual Report of the Ontario Bureau of Mines, vol. XXIX, part I, will be found a full report as to location and occurrence of the Shebandowan Lake nickel deposits.

On August 17th, 1922, five bags were forwarded to the Ore Testing Laboratories, containing five samples of this ore, collected by Dr. T. L. Tanton, and submitted by the Director of the Geological Survey, with a detailed memorandum concerning these samples, as follows:—

Sample No.	Locality	Description	Nature of work required
1	Pit No. 3	Channel sample across 2 feet of ore.	Complete analyses of the samples, and it is requested that concentration and recovery of the platinum groups metals be attempted in those which warrant the treatment.
2	Pit No. 5, east side	Channel sample across 6 feet of ore.	
3	Northerly part of pit No. 10.	Channel sample across 3½ feet of ore.	
4	Southerly part of pit No. 10.	Channel sample across 4½ feet of ore.	
5	Bottom pit No. 10	Picked sample representative of the ore.	

These samples were crushed, carefully sampled down, and were given complete analyses, with the results shown below.

Sample No.	Insol. per cent	Iron per cent	Arsenic per cent	Copper per cent	Nickel per cent	Cobalt per cent	Sulphur per cent	Gold oz./ton	Plat. oz./ton	Pallad. oz./ton
1	16·40	35·65	trace	1·50	5·95	0·20	29·25	0·01	0·04	0·12
2	31·60	26·45	"	3·00	2·10	0·25	18·50	0·007	0·03	0·07
3	38·30	20·70	"	3·10	0·04	0·21	9·10	0·007	0·03	0·10
4	35·90	22·50	"	2·15	0·04	0·21	10·25	0·006	0·02	0·08
5	18·60	33·20	"	5·95	4·10	0·25	27·40	0·083	0·04	0·11

Samples 3 and 4 being very much oxidized and weathered, and furthermore low in nickel, were temporarily discarded. Samples 1, 2, and 5 were mixed, making a combined sample of 40·1 pounds. This was crushed and sampled down and 1,000 gram samples cut out, upon which flotation tests were made, yielding a concentrate, middling, and tailing product. These products, together with a sample cut from the combined head sample yielded the following on analysis:—

Product	Copper per cent	Nickel per cent	Cobalt per cent	Gold oz./ton	Plat. oz./ton	Pallad. oz./ton
Combined head sample	4·78	4·30	0·18	0·06	0·035	0·07
Flotation concentrate	10·77	8·96	0·35	0·10	0·053	0·12
Flotation middling	3·55	3·71	0·21	0·05	0·050	0·09
Flotation tailing	1·77	1·07	0·07	0·026	0·01	0·10

The results from the above concentration tests are not promising. The samples were taken close to the surface where the ore was subject to more or less oxidation, and this may account for the unfavourable results obtained from flotation.

A further shipment of 400 pounds has been received, and it is proposed to conduct further concentration tests, and also to carry out a series of smelting tests to obtain as full information as possible as to the behaviour of the precious metal content.

Test No. 173

THE RECOVERY OF THE VALUES IN A RICH SHIPMENT OF GOLD ORE FROM THE CARIBOU MINING DISTRICT, NOVA SCOTIA

R. K. Carnochan

A shipment of 89½ pounds of gold ore was received at the Ore Dressing and Metallurgical Laboratories from the Herman Hall mine, Caribou, Nova Scotia. This shipment had been sent in by Mr. W. R. Hitchcock, Cornwall, Ont.

The ore consisted of white quartz carrying a very large amount of free gold and small amounts of pyrite and galena. A few specimens were selected from the shipment and the balance, 86 pounds, was crushed in a breaker and a set of rolls to pass 40 mesh. A great deal of metallics was obtained on the screen. The −40 mesh was cut into quarters by means of a Jones riffle and a sample was cut from each quarter for assay. These samples gave the following values:—

	Au oz./ton	Ag oz./ton
−40 mesh, 1st quarter	3·30	0·55
" 2nd "	2·08	0·60
" 3rd "	3·57	0·63
" 4th "	2·71	0·59
Average of four quarters	2·92	0·59

The −40 metallics were cleaned by fluxing in an assay crucible. This gave 2·057 oz. of gold and silver from 86 pounds of ore, which is equal to 47·84 oz. gold and silver per ton. A test on this bullion gave 935·6 parts of gold per 1,000. Therefore the metallics correspond to 44·76 ozs. per ton gold and 3·08 ozs. per ton silver. This added to the average value of the −40 mesh, makes the heads run:—

Gold	47·68 ozs. per ton
Silver	3·67 "

Amalgamation and Cyanidation

Test No. 1.—1,127 grams of the −40 mesh material were mixed in a small pebble jar for 3 hours with 100 grams of mercury and 400 c.c. of water. The contents of the mill were then panned to recover the mercury. The tailing was dried and sampled for assay.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Metallics	44·76	3·08	93·9	83·9
Amalgamated	2·56	0·47	5·4	12·8
Tailing	0·36	0·12	0·7	3·3
Heads	47·68	3·67	100·0	100·0

957 grams of tailing from amalgamation were ground to pass 200 mesh, and agitated for 18 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanided	0·28	0·09	0·5	2·5
Tailing	0·08	0·03	0·2	0·8
Feed	0·36	0·12	0·7	3·3

Amalgamation, Tabling, Flotation, and Cyanidation

Test No. 2.—1,069 grams of −40 mesh material was ground for 3 hours in a small pebble jar with 100 grams of mercury and 400 c.c. of water. The contents of the mill were then panned to recover the mercury. The amalgamation tailing was tabled, making a concentrate and a tailing. The table tailing was floated in a small Ruth machine, making a concentrate and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Metallics		44·76	3·08	47,848	3,293	93·87	83·94
Amalgamated				3,006	448	5·90	11·42
Table conc	28	0·84	3·50	24	98	0·05	2·50
Flot. conc	63	0·52	1·02	33	64	0·06	1·63
Flot. tailing	844	0·06	0·02	51	17	0·10	0·43
Loss	134	0·06	0·02	8	3	0·02	0·08
Heads	1,069	47·68	3·67	50,970	3,923	100·00	100·00

655 grams of flotation tailing were ground to pass 200 mesh and agitated for 33 hours with 6,000 c.c. of water to which had been added 45 grams of sodium cyanide and 10 grams soda ash.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanided	0·06	0·02	0·10	0·43
Tailing	0·00	0·00	0·00	0·00
Feed	0·06	0·02	0·10	0·43

CONCLUSIONS

1. Amalgamation alone on this ore gives a very high recovery. The metallics would be amalgamated in regular milling, so they should be added to the values amalgamated to give the proper recovery by amalgamation. If this is done the recoveries by amalgamation in test No. 1 are gold 99·3 per cent, silver 96·7 per cent, and in test No. 2, gold 99·77 per cent, silver 95·36 per cent.

2. Although the recovery by amalgamation is very high, the amalgamation tailings are too high to discard and some further treatment should be given them.

3. In test No. 1, the amalgamation tailing was cyanided, but this failed to produce a low tailing.

4. In test No. 2, the amalgamation tailing was tabled, floated, and then cyanided. This produced a very low tailing.

5. It is possible that it would be sufficient to table and cyanide the amalgamation tailing.

Test No. 174

THE RECOVERY OF THE VALUES FROM THE ORE OF THE E. H. GLADWIN MINE, BEAVER DAM MINING DISTRICT, NOVA SCOTIA

R. K. Carnochan

A shipment of 5 bags of gold ore, weighing 575 pounds, was received at the Ore Dressing and Metallurgical Laboratories November 13, 1922. This ore was sent in by Mr. W. R. Hitchcock, Cornwall, Ont., and came from the E. H. Gladwin Mine, near Upper Musquodoboit, Beaver Dam mining district, Halifax county, N.S.

The shipment consisted of vein quartz carrying a fair amount of sulphides and some free gold in large flakes. Tests were desired on the ore to determine the best method of extracting the gold.

HEAD SAMPLE

The ore is very difficult to sample due to the presence of metallics. Four samples of about 30 pounds each were taken and crushed through 40 mesh, some metallics being obtained on the screen. The results from these four samples were not satisfactory, so the remainder of the shipment was cut into halves, making two more samples, and these were crushed through 40 mesh, metallics being obtained on the screen. The following table shows the results obtained from the six head samples:—

Sample	Weight lbs.	Metallics				−40 mesh		Total	
		Au mgms	Ag mgms	Au oz./ton	Ag oz./ton	Au oz./ton	Ag oz./ton	Au oz./ton	Ag oz./ton
1.....	33·25	11·04	0·40	0·02	0·00	0·27	0·04	0·29	0·04
2.....	35·00	25·00	1·50	0·05	0·00	0·08	0·00	0·13	0·00
3.....	30·25	338·47	23·33	0·72	0·05	0·08	0·02	0·80	0·07
4.....	30·75	36·46	2·48	0·08	0·01	0·11	0·03	0·19	0·04
5.....	208·00	304·22	20·96	0·10	0·01	0·08	0·04	0·19	0·05
6.....	210·00	298·50	20·50	0·09	0·01	0·01	0·06	0·19	0·07
Total	547·25	1,013·69	69·17	0·12	0·01	0·12	0·03	0·24	0·04

As the table shows, the shipment of ore assays 0·24 oz. per ton gold and 0·04 oz. per ton silver, this being equal to $4.96 in gold and 3c. in silver, making a total of $4.99.

Amalgamation and Cyanidation

A 684 gram portion of the −40 mesh material of No. 1 head sample was amalgamated:—

Product	Au oz./ton	Per cent Au value
Metallics	0·02	6·9
Amalgamated	0·05	17·2
Tailing	0·22	75·9
Heads	0·29	100·0

The amalgamation tailing was ground to pass 200 mesh, and cyanided:

Product	Au oz./ton	Per cent Au value
Cyanided	0·22	75·9
Tailing	0·00	.....
Feed	0·22	75·9

Amalgamation, Tabling and Flotation

A 1,090 gram portion of the −40 mesh material of No. 2 sample was amalgamated, the amalgamation tailing being tabled, and then the table tailing was floated. The flotation concentrate was re-run to clean it.

Product	Weight grams	Au oz./ton	Assay x Wt. gms	Per cent of Au value
Metallics	.....	0·05	54·50	38·5
Amalgamated	.....	.....	34·06	24·1
Table concentrate	34	0·46	15·64	11·0
Flotation concentrate	120	0·26	31·20	22·0
" middling	126	0·05	6·30	4·4
" tailing	681	0·00	0·00	.....
Loss	129	0·00	0·00	.....
Heads	1,090	0·13	141·70	100·0

Tabling and Flotation

A 1,018 gram portion of the −40 mesh material of head sample No. 5 was run over a small Wilfley table, making a concentrate and tailing. The table tailing was then floated in a small Ruth machine, the flotation concentrate being re-run to clean it.

Product	Weight grams	Au oz./ton	Assay x Wt. gms.	Per cent of Au value
Metallics	.....	0·09	91·6	57·8
Table concentrate	26·6	2·52	67·0	42·2
Flotation concentrate	78·0	trace	.....	.....
" middling	121·0	0·00	.....	.....
" tailing	677·4	0·00	.....	.....
Loss	115·0	0·00	.....	.....
Heads	1,018·0	0·19	158·6 193·4	..... 100·0

In this test the table recovers practically all the gold and leaves only a trace in the table tailing. In the amalgamation, tabling, and flotation test, the table did not remove all the gold. These different results are due to the spotty nature of the ore, and the difficulty of getting a representative sample.

Amalgamation and Flotation

Three tests were made on the ore using amalgamation followed by flotation, about 1,000 grams of ore being used in each test. In every one of them more gold was obtained in the products than the amount which assays showed to be in the feed. This is due to the very spotty nature of the ore. These tests show the flotation concentrate to run about 1·25 oz. ton gold, and the flotation tailing about 0·08 oz. ton gold.

CONCLUSIONS

The tests show that the ore does not amalgamate very readily, and to make a good recovery of the gold, amalgamation would have to be followed by some other process.

Amalgamation followed by cyanidation gives a very high recovery of the gold and this seems the best method of treatment.

The ore is low grade, and a considerable tonnage of it would need to be available before a mill to treat it is built.

Test No. 175

AN INVESTIGATION OF THE TAILING AND ROCK DUMPS AT THE GOLD MINES OF NOVA SCOTIA—WITH A REVIEW OF THE PAST MINING AND MILLING METHODS, THE PRESENT STATUS OF THE GOLD INDUSTRY IN THE PROVINCE, AND SOME SUGGESTIONS FOR THE REVIVAL OF THE INDUSTRY

C. S. Parsons

INTRODUCTION

It has been frequently estimated that in the early days of gold mining in Nova Scotia, the mills did not recover over 70 per cent of the gold, and in some cases, not over 50 per cent. This has led to the general belief that the old tailing dumps contain high values in gold. A table was compiled by F. H. Mason in 1898 giving assays of tailings and concentrates from different districts. This table shows that large amounts of gold remained in the tailings. Old operators also report high losses, especially when treating ores containing an appreciable quantity of arsenical pyrite.

The rock or waste dumps are also believed to carry considerable amounts of gold. There would seem to be some foundation for this belief, as the early returns to the Department of Mines show a yield of approximately ¾ oz. of gold per ton of ore milled, while since 1882 there has been a general falling off from this figure. This indicates that in the early days selective mining or sorting was done to obtain only high grade ore. Consequently much material that was then considered too low grade was thrown on the rock dumps, and the large waste dumps that are found about the old mines bear witness to the fact.

At times attempts have been made to re-treat the old tailing dumps, but they have met with failure, except where operating mills were able to re-run their own tailings.

The recent demand for arsenic, with the consequent increase in its price has revived the hope that some of the old tailing dumps could be profitably re-worked for their gold and arsenic content.

To determine the value of the rock and tailing dumps, it was planned to examine and sample as many of them as possible. This would include an approximation of the tonnages available for re-treatment and the assay values in gold, silver, and arsenic. Information was also to be collected on the mining and milling methods, in order to suggest, if possible, some way in which the gold mining industry could be encouraged. Test work was to be done at the laboratories of the Ore Dressing and Metallurgical Division upon samples from the dumps if the preliminary investigation showed that they contained enough gold or arsenic to warrant it.

Through the co-operation of Mr. Faribault of the Geological Survey, the help was obtained of Mr. Cruickshanks, for many years his assistant. Mr. Faribault also provided a list of mines that he considered the most important to visit. This saved much time and was of great assistance.

Visits were made to twenty-one gold districts, where fifty tailing dumps and numerous rock dumps were sampled. The tailing dumps are generally fan-shaped, grading into a swamp, the sands remaining on the slope, the slimes running into the swamp. The depth as a rule does not exceed five feet. They were sampled with a three inch flange auger, but where it could not be used because of the holes caving in too rapidly, a split pipe auger with a removable solid core was used.

To properly sample the rock dumps they should have been trenched, but it was impossible to spare the time necessary for this. All that could be done was to take grab samples from the surface of some of them. Of course, these rough samples cannot be very representative and too much reliance should not be placed on the assay results from them.

It was hoped that the data derived from the examination and sampling of the dumps would shed some light on the past milling practice in the province.

Past Mining and Milling Practice

The mines were small, and the majority of them had scarcely passed the prospect stage. Even the largest do not seem to have milled more than 75 tons per day averaged over a period of a year. Old mining methods have been frequently blamed for helping to cause the decline in the industry, but disparaging the efforts of the old time miners will not revive it. However, it may be well to keep in mind the possibility that some of the mines which are now closed might be reopened and profitably worked.

With a few exceptions very little systematic development was carried on and no assay plans or underground records were kept. Underhand stoping was largely used and but little ore was developed ahead of stoping. The ore shoots were followed closely, but sooner or later lean areas were bound to be encountered. The operators were then forced to abandon work either on account of insufficient funds or because of lack of the necessary underground records by means of which they could possibly have located the extension of the ore shoot.

It may be well to remark that although there has been much useful and instructive structural geological work done on the gold areas, the economic geology based on assays and tonnages has been comparatively neglected.

The mills generally consisted of five, ten and sometimes twenty stamps, and relied wholly upon plate amalgamation for the recovery of the gold. Later, tables were introduced, and installed in most of the mills. The gold in the table concentrate was associated with arsenopyrite and pyrite, and the recovery made was poor. A number of mills stacked their concentrates without attempting to treat them. (A list of table concentrates at some of the mines is given in table I).

At two of the operating mills, samples were taken of the tailings from the amalgamation plates. One of these mills was working on $3 to $4 feed, and five samples taken over a period of five shifts showed 0·02 to 0·05 oz. per ton (average 0·03) of gold in the tailings (see table No. IV). This was a loss of about 60 c. per ton.

The tailings from the other mill were much higher, 0·22 oz. per ton or about $4.40, but it was treating $12 ore. These two mills were working on typical Nova Scotia ores and following the practice that has been in vogue for the past 40 years. Their recoveries should, therefore, approximate those of the older mills.

The Tailing Dumps

The tailing dumps are, with few exceptions, small, so they will have to contain relatively high values of gold and arsenic before it would pay to re-treat them. A list of the dumps sampled is given in table No. II, together with the number of samples taken, the gold, silver, and arsenic content, and approximate tonnages. The tonnages were obtained by a rough measurement of the dumps by compass bearings and pacing. This list shows that none of the dumps sampled contain values worth considering.

Comparing these results with those in the table compiled by Mason, the very surprising point is noticed that the gold that was in the tailings is not there now. If it ever was there, and the tailings from the mills operating at the present time would indicate that it was, it is an interesting problem to determine what has become of it, and how it was taken out.

In volume V (1899-1900) of the Transactions of the Mining Society of Nova Scotia, is a paper by Mr. B. C. Wilson on this very problem. He claims that lavish use of cyanide for dressing the plates gradually leached the gold from the tailing dumps. This explanation is very plausible, as cyanide was used freely, and the dumps being small would be pretty thoroughly saturated by the dilute solvent solution.

A screen analysis of the tailing from one of the mills now operating shows that 50 per cent of the gold is in the −200 mesh product. This fine gold would be readily dissolved, especially when the solution was in contact for such a long time.

There is generally a large amount of decayed vegetable matter at the bottom of the dumps, and it might be expected that part of the dissolved gold would be precipitated there. Some of the samples taken contained considerable amounts of this vegetable matter, but the gold content was not noticeably higher.

The tailing dumps at Montague were not sampled as the Metals Recovery Company of Canada had obtained control of them. It is expected they will be re-treated for their gold and arsenic content, as it is said they assay 2-3 per cent arsenic, and about $3 in gold.

Rock Dumps

A list of samples from a few rock dumps is given in table No. III. As mentioned previously these samples cannot be relied upon to accurately represent the values in the dumps. They are published merely to give some idea of the content. They show that the dumps do not contain much gold or arsenic.

West Gore Antimony Dumps

Samples were taken of the sand tailing and rock dumps of the old antimony mines of West Gore. Table No. V shows the value of these dumps in gold, arsenic, and antimony from which it is seen that the sand dumps contain a fair amount of gold. The slimes from the mill have been run into the river at the foot of the hill. From personal experience in concentrating antimony ore, it is judged that they carried away large losses in gold and antimony. Stibnite ores can be readily concentrated by froth flotation, with the production of a concentrate carrying 50-60 per cent metallic antimony with a recovery of better than 90 per cent. If the gold in the West Gore ores is associated with the stibnite, it would also be recovered by the same treatment.

Flotation tests will be made on the sand tailings to determine if the gold follows the antimony into the concentrate. The successful solution of the problem of concentrating these ores by flotation should encourage prospecting for other antimony veins in that locality.

Increase in Gold Mining During the Past Year

The interest in gold mining is becoming greater, and there has been more work done during the past year than for some years.

Prospecting is being done in the western section of the province at Cranberry Head, Whiteburn, Malaga, Leipsigate, and Brookfield. At Brookfield the King mine is doing some development work and is experimenting with cyanidation under the direction of Mr. Badger. The Malaga Mining Company at Malaga is trying to operate its property but the lack of power is a serious handicap. The use of coal is out of the question and wood would cost $5-$6 a cord, and at best is unsatisfactory.

Mr. S. A. Hister of Halifax is doing work on the Parker-Douglas property in the same district. At Mt. Uniacke Mr. James Crease is running a ten stamp mill using plate amalgamation only. A sample of the tailing running to waste from the plates assayed 0·22 oz. in gold, about $4.50 per ton. This is a very high tailing and indicates that amalgamation will not always save all the gold. A table test on this tailing was made at the Ore Dressing and Metallurgical Laboratories, and a concentrate produced which ran 4·12 oz. per ton with a tailing of 0·06 oz. per ton.

There is a little prospecting going on at South Uniacke and Oldham.

The Bradbury Mines Ltd. at Tangiers were keeping their mine pumped, and in November were waiting for an engineer to make an examination. Since then it is reported that the diamond drilling and other development work recommended by him will be immediately carried out.

At Caribou Mr. Hermon Hall was doing some prospecting and was operating a five stamp mill and a Wilfley table. He had uncovered a number of leads which showed considerable quantities of free gold, but he had done very little underground development and was endeavouring to sell the property as it stood. The concentrate from the table contained 0·85 ozs. gold and 18·43 per cent arsenic, and the tailing contained 0·08 oz. gold and 1·19 per cent arsenic.

Mr. Matthew Higgins was doing a little prospecting at Moose River. A sample of the mineralized slate from the walls of his working ran 0·44 oz. per ton in gold and carried considerable arsenic.

There was some work done at Beaver Dam during the summer, but most of the time seems to have been spent in trying a secret process for the recovery of the gold.

At Goldenville, Mr. J. A. Warner is operating for the Sherbrooke Mining Company and is doing development work and running a small mill of twenty stamps. He is working on low grade ore, but by careful management is making expenses and hopes to put the mine on a paying basis. He is working under ideal conditions as regards power.

At Cochrane Hill work is being done by two separate parties. Mr. Grant McDonald is doing some development work on the Mitchell belt and is blocking out a fair tonnage of ore. This is one of the large belts or series of veins and mineralized slate and quartzite which is worthy of attention as it carries gold values across its width of 75 ft. Mr. Brown has opened up a mine on a large slate belt at the foot of Cochrane Hill. He has also built a mill of his own design for the treatment of the ore.

Suggestions for the Revival of the Industry

The gold industry in Nova Scotia can best be encouraged by giving more attention to improving conditions in the gold areas which limit the successful development and profitable operation of the ore bodies.

The gold is not evenly disseminated through the quartz veins but is segregated in shoots or lenses. This characteristic makes mining uncertain, and unless a systematic geological study is made of underground conditions and a wise policy of development based on geological information, systematic sampling and assaying is carried out, so that ore shoots will be developed ahead of stoping operations, there are small chances of success. The mine operators now in the field should be induced to embark on such a policy.

Very little systematic sampling was done in the past, either underground or in the mill. Where the values are so erratic as is the case with the Nova Scotia gold ore, more of such work should be done to obtain an intelligent idea of the values of the ore being mined, of the mill feed, and of the pulp from each of the milling operations. By encouraging the operators in this respect, it will enable them to detect at once the weak points in their operations requiring attention, and their efforts can be directed to their improvement. It will enable them to ascertain the tonnage of milling ore, the values in such ore, the values in their mill feed, in the pulp from each milling operation, in their tailings. It will also enable them to ascertain their recoveries and the value of the ore to be milled, so that a profit can be made. Such systematic work is essential to successful operation.

The mining laws of the province should be studied to determine if improvements can be made which would encourage more prospecting.

Cheap power for operating purposes should be given consideration. In certain of the mining districts, especially in the western portion of the Province, the cost of fuel, either coal or wood, is a great handicap. The question of operating with oil engines of the Diesel type, or of obtaining cheap hydro-electric power should be investigated to determine the most suitable source of power for the operator.

Research and experimental test work should be carried out on the milling and metallurgical treatment of the ores, to improve the milling practice. Attention should be given to developing a simple and economical method to increase extraction in the small mills, recovering more of the values in the tailings from the plates after amalgamation.

This can be accomplished by cooperation between the provincial and federal mining departments.

The present, when so much attention is being paid to gold mining in Canada, is an opportune time to encourage the industry in Nova Scotia. If the above suggestion can be carried out, the existing mines and other ore bodies which may be discovered in the future, will have a much better chance of being worked as profitable enterprises than has been the case in the past.

TABLE No. I

Concentrates Stacked at Old Mills

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Caribou, H. Hall, conc.	1	0·85	.....	18·43	10	
" " tailing	1	0·08	.....	1·19	.....	
" Lake mine, conc.	1	0·42	.....	trace	100	Cyanided
Cochran Hill, conc.	1	0·60	.....	18·65	10	
Harrington Cove conc.	1	0·05	.....	26·56	few	
Forest Hill, Bendego mine conc.	1	0·20	.....	18·96	20	Cyanided
Dufferin mines, cyanided conc.	1	0·30	.....	17·70	200	
Dufferin mine, conc.	1	0·93	.....	18·96	200	

TABLE No. II

Samples and Assays of Old Tailing Dumps

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Goldboro, Richardson—						
" " No. 1 dump	2	0·10	.....	4·58	10,000	Carried considerable sulphides
" " No. 1 "	6	0·15	.....	1·30		
" " No. 2 "						
" Holes 1–6	6	0·01	.....	0·16	62,000	
" Holes 7–9	3	0·013	.....	0·095		
Goldboro, Skunk den dump	1	0·03	0·01	trace	1,000	
" Gold finch dump	1	0·02	trace	trace	1,500	
Forest Hill, Bendego dump	3	0·01	trace	0·28	25,000	
Wine Harbour, Mapier	1	0·02	0·004	.....	.....	
" Plough Lead	1	0·02	0·004	0·09	1,200	
" Provintial	1	0·02	0·004	0·28	1,500	
Cochran Hill, McDonald	1	0·02	0·008	trace	.....	
Goldenville, Bluenose dump	1	0·04	0·01	0·72	none	
" "	6	0·02	.....	0·19	12,600	Sand
" Royal Oak	2	0·02	trace	0·35	.....	Slime
" McFarlane lake	11	0·02	0·01	0·12	13,500	
" Sherbrook Mining Co.	8	0·025	.....	0·19	180,000	
" " "	3	.....	.....	.....	.....	
Harrington Cove	1	.....	.....	.....	.....	
Dufferin Mine	1	0·03	0·01	0·12	1,000	
" Old Archibauld No. 1 dump	3	0·01	0·003	0·15	3,500	
Moose River, G. & K. Mill No. 2 dump	2	0·03	0·01	0·81	2,000	
" Montreal Mill	3	0·015	0·004	0·28	2,000	
Caribou, Elk Mill	2	0·04	0·01	0·31	1,200	
" Lake Mine	3	0·015	0·003	trace	3,000	Cyanided
" Caffrey Mine	6	0·021	0·004	0·206	20,000	
Tangiers, Brunswick crusher	6	0·02	0·007	0·50	6,000	
" Essex crusher	1	0·02	0·004	trace	.....	
Mt. Uniacke, No. 3 dump	4	0·015	trace	0·22	3,000	
" Great belt No. 2 dump	1	0·01	0·002	0·19	1,000	
" Montreal dump	6	0·02	trace	0·25	8,000	
Lake Catcha, Oxford dump—	6	0·02	0·008	0·34	8,000	
Holes 1–4	4	0·06	0·03	0·10	40,000	Wash from rest of dump, slime
" 6	1	0·02	trace	0·60		
" 5 and 7	7	0·04	0·02	0·12		
Lake Catcha, Anderson dump—						
Holes 1–5	5	0·12	0·02	0·52	10,000	Slimes
" 6	1	0·01	0·01	0·06		
" 7	1	0·04	0·01	0·30		
Lake Catcha, Hanwright mine	1	trace	trace	trace	.....	
Rawdon, McNaughton dump	2	0·021	0·004	0·25	5,500	
Renfrew, Pictou Development Co.—						
Dump No. 1	1	0·015	trace	trace	1,800	
" No. 2	2	0·015	trace	trace	1,800	
Oldham, upper dump, holes 3, 4, 5	3	0·05	0·01	0·57	12,000	
" lower dump, holes 1 and 2	2	0·08	0·01	0·02	3,000	
Waverley, Taylor-Hardman—						
Holes 1 and 2	2	0·01	trace	trace	50,000	
" 3 and 4	2	0·025	0·005	0·89		
" 5	1	0·025	0·005	0·93		
Waverley, Gold Mining Co.	1	0·01	0·003	trace	.....	All in lake
" American dump	2	0·028	0·004	0·54	10,000	
" Wilsons Hall dump	2	0·02	0·001	0·35	1,000	

TABLE No. II—*Concluded*

Samples and Assays of Old Tailing Dumps—Concluded

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Gold River, London mine No. 1.	2	0·01	trace	0·22	2,500	
" " " No. 2.	1	0·02	0·003	trace	.....	
Malaga, Ballow dump, hole No. 1	1	0·017	0·003	0·31		
" " " No. 2	1	0·02	0·005	0·35		
" " " No. 3	1	0·03	0·004	0·60	16,000	Cemented material
" " " No. 4	1	0·022	0·003	0·38		
" " " No. 5	1	0·065	0·012	2·45		
" Caladonia dump.....	3	0·025	0·005	0·206	700	
" Parker-Douglas—						
Hole No. 1.....	1	0·033	0·006	0·412		
" No. 3 and 4.....	2	0·01	trace	0·38	10,000	
" No. 2.....	1	0·025	trace	0·33		
" Malaga Mining Co.—						
Dump A.....	4	0·035	0·003	0·28	3,500	
" B.....	3	0·03	0·006	0·190	700	
" C.....	3	0·025	0·009	0·100	4,000	Sand
" C.....	2	0·060	0·010	0·380		Slime
Brookfield, King mine—						
Hole No. 1.....	1	0·024	0·006	0·285	very little	
" No. 2.....	1	0·032	0·003	0·760		
Whiteburn—						
Dump A, hole 1.....	1	0·005	trace	trace		
" A, " 2.....	1	trace	"	"		
" A, " 3.....	1	0·006	"	"	2,000	
" A, " 4.....	1	0·005	"	"		
" A, " 5.....	1	0·03	0·009	"		
" B, " 1.....	1	0·05	0·02	"	.....	No tonnage
" C, " 1.....	1	0·04	0·02	"		Sand
" C, " 2.....	1	0·025	0·01	"	1,400	Sand
" C, " 3.....	1	0·015	0·003	"		Slime
" C, " 22.....	1	0·015	trace	"		Slime

TABLE No. III

Some Rock Dumps

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Lake Catcha, Oxford.....	1	0·02	0·004	trace	.....	
" Anderson.....	1	0·028	0·007	"	.....	
Dufferin Mine.....	1	0·012	0·003	0·38	.....	
Caribou, Lake mine.....	1	0·01	trace	trace	.....	
" Caffey mine.....	1	0·025	0·003	0·254	.....	
Rawdon, McNaughton.....	1	0·02	0·003	0·19	.....	
" "	1	0·02	0·003	0·174	.....	
Moose River, Torquay.....	1	0·01	trace	1·07	.....	
" G. & K.....	1	0·013	0·002	trace	.....	
Goldenville, Sherbrooke mines..	1	0·028	0·004	0·285	.....	
Mt. Uniacke.....	1	0·02	0·004	0·174	.....	

TABLE No. IV

Samples Taken of Tailings Running to Waste From Operating Mills Using Amalgamation Only

	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Small 10 stamp mill, working on approximately $12 ore	1	0·22	0·04	nil	.....	
Small 20 stamp mill, working on $3-$4 ore—samples from five shifts—						
Sample No. 1	1	0·05	0·08	.....	.....	
" No. 2	1	0·04	0·06	.....	.....	
" No. 3	1	0·02	0·05	.....	.....	
" No. 4	1	0·04	0·06	.....	.....	
" No. 5	1	0·02	0·06	.....	.....	.....

TABLE No. V

West Gore Antimony Tailing and Waste Dumps

Sample	Approx. tonnage	Per cent Sb	Per cent As	Ozs. Au	Ozs. Ag
Sample No. 1, coarse sand tails	1,500	1·30	0·285	0·06	0·01
" No. 2, fine sand tails	14,000	2·30	0·15	0·12	trace
" No. 3, rock dumps (Rock dumps from old shaft.)	27,000	0·38	0·25	0·02	"
" No. 4, rock dumps (From below 800 ft. level.)	27,000	trace	0·396	0·02	"

NOTE.—These sands contain no slimes. The slimes were run into the river and lost. They probably contained very high values in antimony and gold.

Assays of Gold

(Mason)

District	Tailings from plates		Tailings from dumps		Concentrates	
	Maximum	Minimum	Maximum	Minimum	Maximum	Minimum
	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.
Cow Bay	0 1 18	0 – 21	– – –	– – –	– – –	– – –
Sherbrooke	0 3 12	0 – 21	0 6 10	– 3 6	1 8 0	1 8 0
Caribou	0 2 22	0 – 14	0 13 2	– 2 23[1]	12 0 20	0 2 22
Uniacke	0 8 4	0 – 21	0 – –	– – –	13 3 3	1 17 21
Stormont	0 15 8	0 1 18[2]	0 16 22	– 3 6	136 1 19[1]	1 7 0
Brookfield	– – –	– – –	0 13 1	– – 14	12 17 0	1 3 0
Waverley	0 2 15	0 – 21	0 6 12	– – –	4 12 1	1 8 7
Central Rawdon	0 2 22	traces	0 14 17	– 4 16	5 15 0	1 2 4
Wine Harbour	0 3 12	0 1 0	0 6 17	– 3 12	– – –	– – –
Tangier	0 1 4	0 – 21	– – –	– – –	– – –	– – –
Fifteenmile Strm	0 13 10[1]	0 – 7	0 19 20	– – 7	– – –	– – –
Oldham	– – –	– – –	– – –	– – –	3 8 0	1 8 4
Gold River	– – –	– – –	7 12 20[1]	– – –	– – –	– – –
Lunenburg Co	– – –	– – –	– – –	– – –	99 16 18[1]	21 1 8
Montague	8 2 8[3]	0 1 4	0 7 0	– – –	7 11 16	5 9 16

[1] Contained amalgam in appreciable quantities.
[2] The ore was giving only $2 in free milling at time the tailings were taken.
[3] Tailings from ore containing upwards of 40 per cent mispickel.

Test No. 176

CONCENTRATION TESTS ON TAILING SAMPLES FROM TWO MILLS OPERATING ON TYPICAL NOVA SCOTIA GOLD ORES

C. S. Parsons and R. K. Carnochan

A sample of tailings from an amalgamation mill at Mt. Uniacke, Nova Scotia, operated by Mr. Jas. Crease, was received November 9, 1922, at the Ore Dressing and Metallurgical Laboratories.

An assay for gold and silver was desired on the sample and also some experimental work to determine if the values present were unamalgamated free gold and silver, or gold and silver locked up in the sulphides.

The sample as received weighed 360 grams, and upon being dried the weight was 357 grams. This 357 grams was screened on 90 mesh and the oversize was crushed on a muller board by gently rocking the muller, the purpose of this being to roll out any free gold or silver into flakes and not to break up these flakes. The crushed material was screened and the +90 put back on the board and crushed again. This alternate crushing and screening was continued until only a small amount of oversize remained. An assay of this oversize gave only a trace of gold and silver.

Forty grams of the −90 mesh was cut out and crushed in the same manner as described above to pass 200 mesh. The small amount of +200 gave only a trace of gold and silver upon assay. The remainder of the −90 mesh, 310 grams, was floated in a small Janney flotation machine, the concentrate being re-run to clean it. This gave three products from flotation—a concentrate, a middling, and a tailing.

The flotation concentrate was tabled on a small Wilfley table making a concentrate and a tailing. The flotation middling and tailing were mixed and tabled making a concentrate and a tailing. The overflow from the boxes during the tabling of the flotation products was run to a large settling tank and in this way a slime product was obtained. The two table concentrates were so small that they were combined. All products were dried, weighed and assayed.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au oz./ton x grams	Ag oz./ton x grams	Per cent of total values	
						Au	Ag
Table concentrate from flot. prod....	5·05	4·12	1·38	20·81	6·97	30·5	56·2
Table tailing from flot. conc.....	4·25	1·05	0·35	4·46	1·49	6·5	12·0
Table tailing from flot. mid. and tailing.....	78·0	0·05	0·01	3·90	0·78	5·7	6·3
Slime from tank.....	133·0	0·05	0·01	6·65	1·33	9·8	10·7
Loss.....	89·70	0·36	0·02	32·38	1·83	47·5	14·8
Heads.....	310·00	0·22	0·04	68·20	12·40	100·0	100·0

Owing to the small size of the sample submitted and the number of operations carried out on it, no great accuracy can be expected in the above table. Although far from conclusive, the results of the test work indicate that the gold and silver in the sample are locked up in the sulphides and are not present in the free state.

Table Test on Tailing From Same Mill

Although a head sample could not be obtained this mill was treating ore of an average grade of about $12 per ton. A table test was made on the sample of amalgamation tailing which contained 0·22 oz. gold per ton. A concentrate was produced which contained 2·63 oz. per ton with a recovery of 74·5 per cent of the gold. The tailing assayed 0·06 oz. per ton. This mill depended wholly on amalgamation for the recovery of the gold values.

Table Test on Tailing From Mill Operated by J. A. Warner, Sherbrooke Mining and Milling Co., Goldenville

This mill also depended wholly on amalgamation for the recovery of the gold. Five samples taken of the tailing from the battery plates on five different shifts, gave the following assays:

	Gold	Silver
Sample No. 1	0·05	0·08
" No. 2	0·04	0·06
" No. 3	0·02	0·06
" No. 4	0·04	0·06
" No. 5	0·02	0·06

This mill was operating on about $3 ore.

To obtain sufficient material for a table test, the five samples were mixed. A head sample cut from the lot assayed 0·03 oz. per ton gold.

Product	Weight grams	Per cent by weight	Gold			Arsenic per cent
			Assay ozs.	Content ozs. x grams	Recovery	
Concentrate	81	2·02	0·66	53·46	44·6	13·25
Middling	257	6·45	0·04	10·28	8·6	.....
Tailing	3,653	91·53	trace	55·99	46·8	.....
Totals	3,991	100·00	0·03	119·73	100·0	.....

In practice the middling would be returned to the head of the table, and a 50 per cent recovery of the gold in it could be safely figured upon. In this case it would be equivalent to 30 cents per ton of ore. An average of about 40 pounds of concentrate would be obtained per ton of ore treated.

There was not enough concentrate for experiments on the recovery of the gold from the concentrate. Pan amalgamation is suggested as a possible method of recovery.

Test No. 177

THE SEPARATION OF DOLOMITE FROM MAGNESITE IN THE GRENVILLE MAGNESITES

R. K. Carnochan

Shipments of magnesite from the North American Magnesite Company, Calumet, Que., were received at the Ore Dressing and Metallurgical Laboratories as follows:

Feb. 2, 1920	Lot No. 1	84 lbs.	gross	Calcined magnesite
Mar. 26, 1920	" No. 2	5,849 "	net	Raw magnesite
Mar. 23, 1921	" No. 3	24,000 "	"	"

The magnesite contains dolomite in sufficient quantity to make the lime content very high. Magnesites from Austria, Greece, and the state of Washington and California are very low in lime, and are preferred.

The shipments received gave the following analysis:—

Lot No. 1	CaO 23·95 per cent	CO_2	1·22 per cent	Calcined magnesite
" No. 2	CaO 12·85 "	MgO	24·94 "	Raw magnesite
" No. 3	CaO 11·35 "	MgO	35·84 "	"

Lot 1 before calcination would have run about 12 per cent CaO.

Tests were desired on the magnesite to determine if it would be possble to reduce the lime content. The method used to remove the lime consists of three operations:—

1. Calcining the magnesite at a temperature ranging from 950°C. to 1,100° C.

2. Slaking the calcined magnesite with water. The lime slakes into a thin white slurry, while the magnesia remains coarse.

3. Washing out the lime.

Calcining.—Electric, gas, and oil fired furnaces were used to calcine the magnesite. Any one of these furnaces will give a fair product. Care must be taken not to overheat any part of the charge, as this causes clinkers to form. Difficulty was experienced in getting a thorough calcination. However products as low as 1·0 per cent Co_2 were obtained.

Slaking.—The following methods of slaking were tried:

1. Sprinkling the calcined magnesite both when hot and cold with sufficient water to slake, and covering with bagging to keep the heat generated by slaking from escaping.

2. With steam. The calcined magnesite was placed on a screen in a closed vessel, the steam introduced at the bottom passed through the layer of magnesite and escaped around the edges of a wooden cover.

3. With boiling water. The calcined magnesite when cold was thrown into boiling water.

Of all these methods the slaking by means of boiling water gives the best results. The lowest lime content obtained before using this method was 9·0 per cent, but on using it, products running 7·1 per cent lime were obtained.

Washing.—The following methods of washing were used during the test work:

1. Agitating in excess water and decanting, then adding fresh water, agitating and decanting again, this being repeated until all the milk of lime was removed.

2. Stirring in a cylindrical tank with stirring arms close to the bottom, the arms and shaft upon which they operate being hollow, with holes provided in the arms, so that water forced down the shaft enters the tank through these holes, washes the magnesite and overflows at the top of the vessel.

3. Running the slaked magnesite over a Wilfley table, making a sand product and a slime product.

4. Classifying the slaked magnesite by means of a Dorr classifier into sand and slime.

All these methods give the same results. Any one seems to be as suitable as any other.

SUMMARY AND CONCLUSIONS

A washed magnesite product, 7·1 per cent lime, was produced from raw magnesite running 10·7 per cent lime, which had been calcined to 1·2 per cent Co_2. The material washed from the slaked calcined magnesite contains 21·6 per cent lime.

It is desirable to reduce the lime to 3 per cent or less. This has not been done. Starting with magnesite lower in lime than that used and getting a better calcination than that secured in the test work, it would be possible to reduce the lime to less than 7 per cent. It is doubtful if a magnesite product running 3 per cent or less in lime could be made from magnesite similar to that submitted.

Test No. 178

THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE ORE OF THE STIRLING MINE, RICHMOND CO., CAPE BRETON

C. S. Parsons and R. J. Traill

Location of Property.—The Stirling zinc mine is situated in the north-eastern corner of Richmond county, Cape Breton island, half a mile to the south of the Framboise road which connects Loch Lomond with Framboise and Fourchu on the seaboard. The mine is, therefore, in close proximity to points from which the ore could be shipped and to which supplies could be brought by water.

Exploratory Work.—A shaft has been sunk on the ore for 30 to 40 feet. The deposit has been trenched in a number of places, in which can be seen bands of high grade and low grade ore. The property has been drilled by two different companies and some twelve bore holes have been put down.

NOTE.—See Report by D. D. Cairns, Summary Report of Geological Survey, 1916, for further details.

Shipments for Experimental Purposes.—Two shipments of 200 pounds each, one of high grade ore and the other of low grade ore were received at the Ore Testing Laboratories on January 26, 1922. These shipments were submitted by F. M. Connell, Esq., of the Eastern Mining and Milling Company, Toronto, Ont., who had obtained control of the property.

Purpose of Test Work.—It was desired that experimental work be conducted on the two classes of ore to determine methods of concentration and separation of the minerals, and gangue constituents. Saleable products were desired.

Low Grade Ore

The shipment consisted of low grade material, containing the sulphides of iron, zinc, lead, and copper, oxidized by weathering, taken from near the surface or from material which had been broken for some time. It showed on analysis to contain:—

Copper	0·95 per cent
Zinc	3·55 "
Lead	0·76 "
Iron	8·50 "

On examination under the microscope it was found that the minerals were present in a very fine state, and that fine grinding to at least 200 mesh was necessary to free them in order to obtain a separation.

Flotation tests were made, but owing to the oxidized condition of the ore no favourable results were obtained. Table concentration tests made showed that the gangue could be eliminated, with the production of a sulphide product the equivalent of the higher grade ore.

On account of the oxidized condition of the ore, the sample was not considered to be a true representative one and extensive experimental test work was not conducted.

Higher Grade Ore

The shipment consisted of the sulphides of zinc, lead, copper and iron representing 80 per cent and a siliceous gangue of about 20 per cent. It showed on analysis to contain:—

Copper	2·80 per cent	Magnesia	0·63 per cent
Zinc	27·60 "	Soluble iron	0·06 "
Lead	8·10 "	Manganese	trace, less than 0·1 per cent
Iron	11·50 "	Gold	0·08 oz.
Lime	0·26 "	Silver	6·58 "

Examination of polished surfaces of this massive sulphide ore under the microscope showed the minerals to be in a very fine state, closely associated with each other. In comparison with the ore of the Sullivan mine, Kimberley, B.C., where grinding to 200 mesh is practiced to obtain a good separation, this ore showed that even finer grinding is necessary.

Flotation tests were conducted on the raw ore, and on the ore after it was given a slight roast. Leaching tests in a weak solution of sulphuric acid were also conducted on the roasted ore.

Flotation Tests on Raw Ore.—The ore was crushed to pass a 200 mesh screen. Some fourteen tests, namely tests Nos. 1 to 8, and 13 to 18 inclusive, were conducted using various reagents and combinations to effect a separation of the minerals. No results worthy of note were obtained. This may have been due to the oxidized condition of the sample. Further experimental work on the freshly broken ore would be necessary to determine if flotation could be applied to the raw ore.

Flotation Tests on Roasted Ore

A number of tests were made on the roasted ore with encouraging results, as follows:—

Test No. 9

The ore was crushed to 20 mesh and given a slight roast below 450 C. The roasted ore was then ground in a ball mill to 200 mesh and floated with K.K. fuel oil and pine oil with about 15 lbs. of acid per ton or ore. 1,000 grams were used for the test. The results were as follows:

Product	Weight grams	Analysis per cent Zn	Content grams Zn	Percentage of Zn values
Concentrate	580	43·50	252·1	81·7
Middling	110	23·40	25·7	8·3
Tailing	305	10·10	30·8	10·0
Heads	995	31·00	308·6	100·0

Test No. 10

Test No. 9 was duplicated at a temperature of 340 to 370 C. Time in furnace was 6 minutes. The ore was crushed to 30 mesh before roasting. The same procedure for flotation was followed as in test No. 9, 905 grams of ore being used for this test, with the following results:

Product	Weight grams	Analysis per cent Zn	Content grams Zn	Percentage of Zn values
Concentrate	592	41·95	247·2	92·1
Middling	105	13·75	14·4	5·3
Tailing	198	3·50	6·9	2·6
Heads	895	30·00	268·5	100·0

Leaching Tests on the Roasted Ore

A small 100 gram sample was crushed to 20 mesh and roasted for 6 hours at a temperature below 700° C. The roasted ore was leached with 10 per cent hot H_2SO_4. The recovery of the zinc in solution was 90 per cent.

A sample of 50 pounds was crushed to 20 mesh and roasted in a large muffle furnace. The temperature range was 300°C. in the top muffle, to 700° C. in the lower muffle. 95 per cent of the zinc was soluble in acid.

A small sample was roasted in a laboratory muffle furnace for 1½ hours at temperature 550° to 700° C.

Water soluble.....	Zinc 0·9 per cent of total....	— 0·26 per cent
Acid soluble in 10 per cent H_2SO_4	Zinc 99·3 " "	—27·40 "
" 10 " "	Cu 95·4 " "	— 2·67 "

A small sample was roasted in a laboratory muffle furnace for 4 hours at temperature 300° to 700° C.

Water soluble.....	Zinc 2·5 per cent of total...	— 6·10 per cent
Acid soluble in 10 per cent H_2SO_4	Zinc 100 " " ...	—27·60 "
" 10 " "	Cu 94·3 " " ...	— 2·64 "

SUMMARY AND CONCLUSIONS

Due to the oxidized condition of the sample of low grade ore submitted, a separation of the minerals in this class of ore could not be obtained. Table tests show, however, that the gangue material can be eliminated and the minerals constituents collected in a concentrate approaching the grade of the higher grade ore.

Flotation tests on the higher grade ore gave no satisfactory results on the raw ore. This, however, is not conclusive proof that a separation of the minerals could not be obtained on freshly broken ore.

Flotation tests on the higher grade ore after it was given a slight roast gave very satisfactory results. A zinc concentrate containing 45 per cent zinc was obtained by this method.

Leaching tests on the roasted higher grade ore showed the zinc and copper content readily soluble in weak sulphuric acid.

Test No. 179

CONCENTRATION TESTS ON LE ROI No. 2 DUMP ORE

C. L. Dewar

A shipment of 100 pounds of Le Roi No. 2 dump ore was received February 21, 1922, from Douglas Lay, Esq., Rossland, B.C. The shipment was submitted in connection with our investigation on the use of Canadian manufactured reagents for flotation of Canadian ores.

On analysis the ore was found to contain:—

Copper	0·52 per cent
Iron	8·90 "
Gold	0·16 oz. per ton
Silver	0·18 "

A number of tests were conducted using various reagents, the results of which are given in the following tables:—

Concentration Tests on Le Roi No. 2 Dump Ore

LE ROI No. 2 MINE—FLOTATION OF RAW ORE

Test No.	Product	Weight per cent	Analysis		Recovery per cent		Remarks
			Cu per cent	Au oz.	Cu	Au	
1	Concentrate	13·4	3·65	0·70	92·0	57·5	Coal tar creosote 1·0 lb./ton
	Tailing	86·6	0·05	0·08	8·0	42·5	Pine oil (G.N.S. No. 5) 0·15 "
							Lime 4·0 "
2	Concentrate	3·3	11·60	1·26	78·8	28·6	
	Middling	5·7	1·00	0·21	11·7	8·2	No. 34 Fuel oil 0·7 lb./ton
	Tailing	92·0	0·05	0·10	9·5	63·2	F.P.L. No. 25 0·4 "
4	Concentrate	2·9	14·20	1·80	78·3	29·0	Lime 5·0 lb./ton
	Middling	9·4	0·75	0·24	13·4	12·5	Mix. "A" 0·7 "
	Tailing	87·7	0·05	0·12	8·3	58·5	G.N.S. No. 28 0·2 "
6	Concentrate	6·3	7·28	1·38	93·1	66·8	Mix. "A" 1·2 lb./ton
	Middling	11·7	0·15	0·16	3·6	14·4	
	Tailing	82·0	0·02	0·03	3·3	18·8	(Deep froth)
7	Concentrate	23·0	2·10	0·50	96·9	79·0	Wood tar, 50 per cent {
	Tailing	77·0	0·02	0·04	3·1	21·0	Heavy wood oil No. 30, 50 per cent } 0·8 lb./ton
9	Concentrate	7·7	5·60	.....	79·6	.....	LeRoi mix. { No. 34 fuel oil / water gas tar
	Tailing	92·3	0·12	.....	20·4	.....	H. W. creosote and P.T.T. No. 350
10	Concentrate	3·3	10·05	.....	67·8	.....	Mix. "A" 1·2 lb./ton
	Tailing	95·2	0·15	.....	27·9	.....	
	Clean up	1·5	1·45	.....	4·3	.....	(Callow cells)
11	Concentrate	14·3	3·35	1·02	91·8	85·0	Mix. "A" 2·3 lb./ton
	Tailing	85·7	0·05	0·03	8·2	15·0	H_2SO_4 10·0 "
12	Concentrate	19·3	2·45	0·56	90·7	65·7	H_2SO_4 10·0 lb./ton
	Tailing	80·7	0·06	0·07	9·3	34·3	Wood tar mix. 0·8 "
							P.T.T. No. 350 0·05 "
17	Concentrate	3·0	7·99	1·56	54·5	26·9	Ground with mix. "A" 2·0 lb./ton
	Tailing	97·0	0·21	0·13	45·5	73·1	H_2SO_4 10·0 " Made very rough concentrate, cleaned with fizzol, midds put in with tailing.

Mix. "A"—Coal tar creosote 60 per cent. Both these from Dominion Tar and Chemical Co.
Coal tar..... 40 "

LE ROI No. 2 MINE—FLOTATION AND TABLING OF RAW ORE

Test No.	Product	Weight per cent	Analysis		Recovery per cent		Remarks
			Cu per cent	Au oz.	Cu	Au	
3	Table conc....	9·4	1·38	0·36	25·0	20·5	2,000 gm. −14 mesh tabled, Table tails and midds ground; floated with K.K. oil 0·5 lb./ton
	Flot. conc.....	2·3	11·45	2·04	50·5	28·5	
	" midd....	5·5	0·65	0·18	6·7	6·0	
	" tails.....	74·2	0·08	0·10	11·3	45·0	
	Table slime...	8·6	0·40	.....	6·5	.....	G.N.S. No. 28 0·6 "
							F.P.L. No. 31 0·1 "
							P.T.T. No. 350 0·2 "
5	Table conc....	17·5	1·13	0·40	38·8	36·6	1,500 gm. −14 mesh tabled. Midd. retabled. Conc. and midd. mixed. Tails reground, floated with mix. "A" 1·4 lb./ton
	Flot. conc.....	4·9	5·60	1·20	53·8	30·8	
	" midd....	9·5	0·02	0·23	0·4	11·4	
	" tails.....	67·5	0·05	0·06	6·6	21·2	
	Table slime..	0·6	0·37	.....	0·4	.....	G.N.S. No. 28 0·25 "
							P.T.T. No. 400 0·4 " (Very deep froth)

FLOTATION AND TABLING OF ROASTED ORE

16	Table conc....	2·8	0·38	0·60	2·1	26·9	1,200 gm. −18 mesh tabled. Tails reground with W.G. tar 1·8 lb./ton and floated with P.T.T. No. 350 0·2 lb./ton.
	Flot. conc.....	1·8	1·12	2·32	3·9	30·0	
	" midd....	10·7	0·55	0·12	11·4	9·2	
	" tails.....	84·7	0·50	0·08	82·6	48·7	

FLOTATION OF ROASTED ORE

13	Concentrate..	4·9	.....	0·95	.....	47·3	Roasted 6 minutes at 800° F.
	Midds.....	8·3	.....	0·31	.....	26·2	Ground with mix. "A" 1·8 lb./-ton
	Tailing.....	86·8	.....	0·03	.....	26·5	P.T.T. No. 350, 0·1 lb./ton
14	Concentrate..	8·4	.....	0·49	.....	39·1	Ground with mix. "A" 1·2 lb./-ton
	Tails.....	91·6	.....	0·07	.....	60·9	
15	Concentrate..	6·4	.....	1·00	.....	?	Ground with mix. "A" 2·5 lb./-ton
	Tailing.....	93·6	.....	trace	.....	?	Floated with P.T.T. No. 350 0·1 lb./ton

CONCLUSIONS

From the results of the above tests, reagents of Mixture "A" in an acid circuit seem to give the better results. The results, using these reagents, are given under Test No. 11 on raw ore. Satisfactory results were not obtained on the roasted ore.

Test No. 180

THE USE OF FLOTATION REAGENTS MANUFACTURED IN CANADA

C. S. Parsons and C. L. Dewar

This investigation was undertaken at the request of Canadian manufacturers of flotation oils and other flotation reagents. The object was to encourage the use of the Canadian products in preference to those imported from foreign countries.

There are only five sources of flotation reagents in Canada, namely, the petroleum industry, the distillation of coals, the distillation of hardwoods, the Canadian Electro Products Co., and the pulp industry where the sulphite process is used and from which a reagent is obtained possessing high frothing qualities.

There is no commercial source of resinous wood oils in Canada. If it is necessary to use oils of this nature, such as pine oils, they must be imported.

In order to obtain information on the consumption of the various reagents used for flotation purposes and their source of supply, the mining companies using flotation in the treatment of their ores were circularized and asked to fill out a questionnaire. On the whole these were filled out and returned promptly, but the accuracy of some of the figures supplied seems to be open to doubt, and it has been necessary to put our own interpretation on them in certain cases. The tables following are compiled from the answers received. A table is also given, compiled from the records of the Mineral Resources Division showing the consumption of oils for the year 1920. The quantity of oils used for flotation probably reached a maximum in that year.

Pine oil makes up 26 per cent of the total oils imported into Canada for flotation purposes. It has been found that with few exceptions, the use of pine oils is not necessary, and other less expensive oils and reagents will give equally good results. The chief instances of where pine oil is essential are in the flotation of molybdenite and graphite ores. Steam distilled pine oils are the only known oils to give satisfactory results in frothing the pulp in the presence of kerosene oil which is used as the collector in the case of these minerals.

There is at present on the market a Canadian flotation reagent which has a frothing power far greater than pine oil, and which can be used even in the presence of kerosene. This reagent is not an oil, has no collecting power for sulphides, and from the results of experimental work with it, can be used in place of steam distilled pine oil in the flotation of practically every ore where pine oil is used. The reagent is known as Fumol No. 6, manufactured by the Canadian Electro Products Co., Shawinigan Falls, Que. It was tried on the flotation of the porphyry copper ores in the United States in one of the largest mills, and found to be equally as satisfactory as pine oil.

It is one thing to recommend the use of certain reagents for those already in use, but it is a very difficult matter to persuade mill men to use them. The reasons are obvious. The operators become familiar with the characteristics displayed by certain oils in the flotation cells, and can tell at a glance how the cells are working. In 1916 this Department,

with the co-operation of the Forest Products Laboratories undertook to find a substitute for pine oil for the flotation of the Cobalt silver ores. This was done at the urgent request of the Cobalt mining companies. The reasons given were that they were unable to obtain an adequate supply of imported pine oil. A substitute was found, namely, hardwood creosote, manufactured in Canada by the Standard Chemical Company of Montreal. To demonstrate finally and conclusively that this oil could be used with equal success, the use of the Buffalo Mill at Cobalt was obtained through the co-operation of Mr. T. R. Jones. This mill was run, using hardwood creosote oil under standard operating conditions, for two weeks, during which time no difficulties were encountered, and the recovery and grade of concentrate was as good as obtained with steam distilled pine oil. In the meantime, the General Naval Stores of New York had obtained long term contracts with most of the southern producers of pine oil and came forward with the guarantee that they would be able to meet the requirements of the mining companies at Cobalt. Not one of the mills ever used the hardwood oils, and even to-day, when pine oil is costing double what it did at that time, and hardwood oil can be obtained considerably cheaper, they have not made the change.

It will be noted that in 1921, 2,800 gallons of hardwood creosote was imported into Canada from Cleveland Cliffs, United States. The Standard Chemical Company, the only producer of hardwood oils in Canada, produces hardwood oils which, in our opinion, have superior flotation qualities to the Cleveland Cliffs product. In our flotation laboratories we have been able to obtain as good results with the Canadian hardwood oils as with the Cleveland Cliffs imported oils, and we recommend their use in preference to the imported hardwood oils.

Samples of 100 pound lots of ore were received from the following companies using flotation, and the test work conducted during the year on these samples is given elsewhere in the Summary Report under their respective test numbers. The work on the remainder will be included in the Summary Report for 1923.

Le Roi No. 2 Limited, Rossland, B.C.	Test No. 179
Consolidated Mining & Smelting Co. of Canada, Trail, B.C.	" " 187
Canada Copper Corporation, Allenby, B.C.	" " 188
Galena Mining and Milling Co., Silverton, B.C.	" " 189
Belmont Surf Inlet Mines Ltd., Surf Inlet, B.C.	" " 190

A list of Canadian mills which our records show use flotation is as follows:—

CONCENTRATION PLANTS IN CANADA USING OIL FLOTATION

Copper Ores— *Name of Company*	*Location*
Eastern Mining and Milling Co.	Eastman, Que.
Eustis Mining Co. (under erection)	Eustis, Que.
Belmont Surf Inlet Mines, Ltd.	Surf Inlet, B.C.
Britannia M. & S. Co., Ltd.	Britannia Beach, B.C.
Canada Copper Corporation	Allenby, B.C.
Consolidated Mining and Smelting Co. of Canada, Trail, B.C.	
Granby Consolidated M. S. & P. Co. (under erection)	Anyox, B.C.
Highland Valley Mining and Development Co.	Highland Valley, B.C.
Kamloops Copper Co	Kamloops, B.C.
Tidewater Copper Co	Sydney Inlet, B.C.

CONCENTRATION PLANTS IN CANADA USING OIL FLOTATION—*Concluded*

Name of Company	*Location*
Copper-Nickel Ores—	
Mond Nickel Co.	Conniston, Ont.
Lead-Zinc Ores—	
Zinc Company, Ltd.	Notre-Dame des Anges, Que.
Consolidated M. & S. Co. of Canada (under erection)	Kimberley, B.C.
Consolidated M. & S. Co. of Canada	Trail, B.C.
Cunningham, Clarence (Slocan ores)	Alamo, B.C.
Galena Mining and Milling Co.	Silverton, B.C.
Noble Five	Cody, B.C.
Ottawa Mine	Slocan City, B.C.
Roseberry-Surprise Mines Co., Ltd.	Roseberry, B.C.
Silversmith Mines, Ltd.	Sandon, B.C.
Standard Silver-Lead Mining Co., Ltd.	Silverton, B.C.
Van Roi Mining Co.	Silverton, B.C.
Gold Ores—	
Premier Gold Mining Co.	Stewart, B.C.
Argonaut Gold Mines, Ltd.	Dane, Ont.
Silver Cobalt Ores—	
Bailey Silver Mines, Ltd.	Cobalt, Ont.
Cobalt Provincial Mining Co., Ltd.	" "
Coniagas Mines, Ltd.	" "
Dominion Reduction Co.	" "
McKinley-Savage-Darragh Mines, Ltd.	" "
Mining Corporation of Canada	" "
" " " Buffalo Mill	" "
M. J. O'Brien, Ltd. (Bailey Mill)	" "
National Mines, Ltd.	" "
Peterson Lake Silver-Cobalt Mining Co., Ltd.	" "
Molybdenum Ores—	
Dominion Molybdenite Co.	Quyon, Que.
St. Maurice Mines, Ltd.	Indian Pen., Que.
	Amos, Que.
Molybdenum Products Co.	Wilberforce, Ont.
Renfrew Molybdenum Mines, Ltd.	Mt. St. Patrick, Ont.
Steel Alloy Corporation	Dacre, Ont.
Molybdenum Mining and Reduction Co.	Alice Arm, B.C.
Graphite Ores—	
Black Donald Graphite Co.	Calabogie, Ont.
Globe Graphite Mining and Refining Co., Ltd. (The)	Port Elmsley, Ont.
National Graphite, Ltd.	Harcourt, Ont.
Timmins Graphite Mines (The)	Westport, Ont.
Consolidated Graphite M. & M. Co.	Buckingham, Que.
Laurentian Graphite Co.	St. Rémi, Que.
North American Graphite Co., Ltd. (under erection)	Buckingham Que.
Quebec Graphite Co., Ltd. (The)	"
Standard Graphite Co.	Guenette, Que.

TOTAL OILS USED IN FLOTATION

1921-22

Oil	Quantity used	Pounds of oil used per ton of ore
	Pounds	
Coal tar creosote	223,120	0·44
Coal tar	123,750	0·25
Pine oil, G.N.S., No. 5	8,400	0·035
Pine oil, P. T. & T. Co., No. 350	56,270	0·28
Water gas tar	153,750	0·50
Solvent naphtha and pitch	5,760	4·0
Hardwood creosote oils	28,250	0·25
Total	599,320	

OILS IMPORTED FOR FLOTATION

Oil	Quantity
	Pounds
Pine oil, steam distilled, G.N.S. No. 5	8,400
Pine oil, Pensacola Tar and Turp. Co., No. 350	56,270
Cleveland Cliffs Hardwood Oils	28,250
Water gas tar	153,750
Total	246,670

Quantity of ore reported treated by flotation, 1921–1922	617,844 tons
Quantity of oil used	599,320 lbs.
Quantity of oil used per ton of ore	0·97 lbs.

ADDITIONAL REAGENTS

Reagent	Pounds	Average cost per pound	Total cost	Ore
			$	
Copper sulphate	303,750	3 to 10c.	24,375	Lead-zinc
Soda ash	1,800,000	3·3c.	59,400	Lead-zinc
Sodium silicate	11,000	.....	330	Silver and copper
Sulphur	3,920	3·0c.	117	Cobalt-silver
Sod. sulphide	2,600	4·6c.	120	Copper, gold

FLOTATION OILS CONSUMED IN 1920

—	Quantity gals.	Average cost per gal.	Total cost
			$ cts.
Oil imported	116,173	49·0c.	57,219 40
Oil domestic	101,270	24·6c.	24,665 81
Total oil consumed	217,443	37·7c.	81,885 21

Of the above total 5 per cent of oils used were imported.
Of the above total 132,410 gals. were used in the Cobalt district.

Tests Nos. 181 and 182

THE METALLURGICAL TREATMENT OF CANADIAN PYRRHOTITE ORES BY LEACHING AND ELECTROLYTIC DEPOSITION, AND OF CANADIAN PYRITE ORES BY ROASTING, LEACHING, AND ELECTROLYTIC DEPOSITION, WITH THE PRODUCTION OF ELECTROLYTIC IRON, AND THE RECOVERY OF BY-PRODUCTS SUCH AS SULPHUR, COPPER, ETC.

R. J. Traill

INTRODUCTORY

An investigation to determine the suitability of a leaching and electrolytic method of treatment for pyrrhotite and pyrite ores in connection with their possible utilization in the manufacture of electrolytic iron, was started in November 1922.

The preparation of electrolytic iron has been a subject of scientific study and research for about 80 years, but it is only during the last 15 or 20 years, with the realization of its unusual physical and chemical properties, that metallurgists and chemists have turned their attention toward establishing for it a practical industrial process. Processes are in operation at the present time in France, Germany and America, producing an iron of 99·9 per cent to 99·97 per cent purity, from pig iron or scrap used as anodes in an electrolyte of ferrous salts, with and without additional reagents, on steel cathodes of either sheet or rotating mandrel form.

Electrolytic iron has unusual mechanical and electrical properties which open up many special fields of usefulness, and it is in demand in the making of special steel or alloys, boiler tubes, electrical machinery, wire, and articles to be nickel plated or galvanized, etc.

The Eustis Process

F. A. Eustis of Boston and C. P. Perin of New York have patented what is known as the Eustis process, with the idea of utilizing iron ores directly. It differs from preceding processes in that an insoluble anode is used, and a diaphragm separates the anolyte and catholyte. In the cell iron is deposited on the cathode, while oxidation of the ferrous chloride takes place at the anode. The ferric chloride thus formed in the anode compartment is led over the raw pyrrhotite ore (pyrite requires special treatment) effecting the solution of the iron and simultaneously reducing the ferric chloride to the ferrous state. Sulphur is deposited in the gangue and is recovered as brimstone. The chemistry of the cell reaction may be shown as

$3FeCl_2$	=	Fe	+	$2FeCl_3$
Ferrous chloride		Iron		Ferric chloride

The production of ferric chloride is directly proportional to the amount of iron deposited.

The reaction of the ferric chloride and ore may be shown by the equation:

$2FeCl_3$	+	FeS	=	$3FeCl_2$	+	S
Ferric chloride		Mono-sulphide or iron		Ferrous chloride		Sulphur

From these equations it will be seen that exactly the amount of iron removed by electrolysis is restored to the solution, and the process becomes cyclical.

In an ore containing copper the copper dissolved must be removed, probably by precipitation on scrap iron, and can be readily recovered as a by-product. It is claimed for the process that 90 per cent of the iron may be recovered as pure electrolytic iron, and 90 per cent or more of the sulphur as brimstone. The current density may be varied over a wide range, 20-100 amp. per sq. ft., and the voltage from 1-4·5 volts.

The Estelle Process

The Estelle process, which in some respects is similar to the above, has been patented by Axel Estelle.

In this process the suitably prepared ore is heated with a weak non-oxidizing acid, preferably hydrochloric, at a temperature of 40-80° C. The result of the reaction is the formation of ferrous chloride and sulphuretted hydrogen. The ferrous chloride goes to the electrolytic cell where a portion of the iron is deposited, and twice this amount of iron is converted to ferric chloride. The sulphuretted hydrogen set free in the acid-ore reaction is collected in absorption towers and used for the reduction of the ferric chloride from the cell, hydrochloric acid being set free with separation of sulphur. The hydrochloric acid previously combined with the non-electrolytically deposited is in this manner again liberated and made available for leaching fresh ore.

It is claimed for this process that other "sulphur containing metals" present remain with the gangue, and can be readily recovered in this concentrated condition, also that the sulphur is eliminated in a separate apparatus in a fairly pure condition.

Laboratory Investigations

The laboratories of the Ore Dressing and Metallurgical Division plan to investigate the possibilities of each of these processes, and is at the present time conducting tests on the Eustis process. The work done so far has been of a very preliminary nature and only on a small laboratory scale. The following brief description of tests are submitted to demonstrate the possibilities of the leaching process as applied to pyrrhotite and pyrite ores.

General procedure followed

An enamel pail of 5-6 litres capacity, fitted with a wooden cover, was used as a leaching tank. Agitation was accomplished by a rotating wooden agitator. Steam was used for maintaining the temperature necessary for leaching. The ferric chloride used was ordinary grade, containing about 20 per cent iron.

It was found that steam passing into a cold or slightly warm solution of ferric chloride hydrolyzed the ferric chloride to an appreciable extent, and the iron so hydrolyzed would be lost as oxychloride of iron in the gangue. By passing the steam into a hot solution, or a slightly acid solution, the hydrolytic dissociation was greatly reduced. Heating directly with steam causes a dilution of solution due to condensation of the steam,

and the increase in volume has been as much as 18 per cent in two hours running on a 3 litre test. This problem of heating the solution will probably be more readily overcome in a continuous circuit process, as the electrolyte will be hot, due to the action of the current. It may also be observed here that the reaction between the ore and the ferric chloride is exothermic to the extent of 20,400 calories or 700 B.Th.U. per pound of iron dissolved.

For laboratory tests a steam jacket method of heating was found most satisfactory; there was a loss of course in volume due to evaporation amounting to 5-15 per cent, but the hydrolytically dissociated iron was greatly reduced.

According to theory, in the leaching process 2 parts of iron as ferric chloride should dissolve 1 part iron from the ore, resulting in the formation of 3 parts iron as ferrous chloride.

Test No. 181

A shipment of pyrrhotite ore received in November from the Smith property, west side of lake Memphremagog, Que., was used in the following tests. This ore on analysis was found to be about 80 per cent pyrrhotite, 25 per cent chalcopyrite, 2·5 per cent lime (CaO). The iron content averaged 49·5 per cent, and the copper 0·85 per cent.

Leaching Test No. 1.—Quantities used:

3·0 litres ferric chloride at 70 grams per litre = 210 grms. Fe.
235 grams ore (−200 mesh at 49·5 per cent Fe = 116·5 grms Fe.

This amount of ore would be about 11 per cent in excess of theoretical requirement.

The ore and solution were put in a pail and heated by passing steam into the solution, agitating at the same time, until the temperature had reached 95° C. which temperature was maintained for 2 hours. The solution was allowed to cool and was then filtered, increase of volume being 18·3 per cent. Theoretically the 210 grams of Fe in the ferric chloride should react with 105 grams of Fe in the ore to form 315 grams of ferrous iron.

Analysis of solution showed that 252 grams of ferrous iron had been formed, which is 80 per cent of theoretical amount, or in other words 84 grams of iron were leached out of the ore. Had the time of leaching been extended, it is probable that a much higher recovery would have been made.

Leaching Test No. 3.—This test was run to obtain data concerning the rate of reaction when using a high excess of ore, and to obtain some information on counter-current extraction, and also on the behaviour of copper in the ore. Quantities used:—

3·5 litres ferric chloride at 66 grams per litre = 231 grams Fe.
335 grams ore (−200 mesh) at 49·6 per cent Fe = 165·8 grams Fe.
and at 0·85 per cent Cu = 2·85 grams Cu.

This amount of ore would be about 45 per cent in excess of theoretical.

The solution was heated to 75°C. by steam jacket method, ore added, and the temperature increased to and held at 95°C.

Solution sampled after 30 minutes leaching contained 85 g/l ferrous Fe.
Solution sampled after 60 minutes leaching contained 92·5 g/l ferrous Fe.
Solution sampled after 90 minutes leaching contained 105 g/l ferrous Fe.
Solution sampled after 120 minutes leaching contained 106 g/l ferrous Fe.
Reaction considered complete, no ferric iron present. Time 2 hours.
Volume of solution after cooling and filtering = 3·10 litres
Analysis showed total ferrous iron = 328·6 grams

Theoretically 231 grams of Fe in the ferric chloride would react with 115·5 grams Fe in the ore to give 346·5 grams of ferrous iron, so that 328·6 grams ferrous iron found is equal to 95 per cent of the theoretical, or 109·5 grams of iron were leached from the ore.

The residue weighed 206 grams and contained 30·6 per cent Fe or 63 grams. Theoretically Fe in residue should be the amount of iron in quantity of ore used, minus amount of iron dissolved in leaching, which would be 165·8 — 109·5 = 56·3 grams. It was found, however, that ferric chloride hydrolyzed and oxychloride of iron precipitated, hence this excess of 6·7 grams in the residue is probably due to (1) iron hydrolytically dissociated, and (2) iron precipitated by lime present in ore.

The copper in residue amounted to 1·17 per cent or 2·41 grams, therefore the amount of copper leached was 15·3 per cent.

Leaching of residue:—180 grams of this residue containing 55 grams Fe and 2·106 grams Cu were treated with 3·4 litres solution containing 45 grams/litre ferrous Fe = 153 gr. Fe. containing 32 grams/litre ferric Fe = 108·8 gr. Fe.

Conditions of leaching similar to first part of test. After 2½ hours leaching analysis of solution showed a content of 73 grams/litre ferrous Fe; and 17 grams/litre ferric Fe.

Volume of solution after cooling and filtering = 3·125 litres. Total ferrous iron is 228·3 grams and ferric iron 53 grams.

The result shows that 25 grams of Fe in the residue has been leached out.

The copper content of the residue was 0·65 per cent, weight of residue 150 grms.

Combining the results of these two tests it is found that:

Total extraction of iron = 83·4 per cent
Total extraction of copper = 61·6 per cent

Summing up it may be said that when a large excess of ore is used the reaction is very rapid and a completely reduced ferrous chloride solution is produced. In practice this reaction would take place in the primary agitators. In the treatment of the residue from this first action, which would take place in the secondary agitators, the reaction would be much slower, due possibly to the liberated sulphur forming a protective coating over the ore particles.

In the above test a higher extraction would probably have resulted had the period of extraction in the secondary action been extended.

As a result of the several tests made, it has been found that fine grinding of the pyrrhotite is essential and that the leaching process should be conducted at a temperature around 90-95° C. Best results have been obtained by grinding to minus 200 mesh (·0029 inch opening) and leaching at 95° C.

Pyrite Treatment

Pyrite (FeS_2) is not readily attacked by ferric chloride, and requires preliminary treatment. This treatment consists of converting the pyrite into a variety of pyrrhotite, soluble in ferric chloride, by roasting in a neutral or reducing atmosphere at a temperature of 700-750° C. Under these conditions one atom of sulphur is driven off and with suitable apparatus can be recovered as a by-product.

Test No. 182

A shipment of pyrite received November, 1922, from the Eustis mine, Eustis, Quebec, was used in the following test:—

Chemical analysis of ore: iron 41·2 per cent; copper 1·55 per cent; lead 0·75 per cent; sulphur 46·7 per cent. This would be about 85 per cent pyrite and 4·5 per cent chalcopyrite. Screen analysis showed:—

—	Sample weight	Per cent	Per cent accumulative
Held on 35 mesh (·0164" opening)	14	7	7
" 48 " (·0116" ")	28	14	21
" 65 " (·0082" ")	40	20	41
" 100 " (·0059" ")	48	24	65
" 150 " (·0041" ")	28	14	79
" 200 " (·0029" ")	28	14	93
Passing 200 " (·0029" ")	14	7	100
	200	100	.....

Pyrite roasting.—Three samples of 150 grams each were placed in a piece of wrought iron pipe 8" x 1½", closed at one end with a cap. These were placed in a small electric muffle furnace and roasted at a temperature ranging from 700 to 750 C. An examination after 45 minutes roasting showed incomplete roasting, particles of pyrite being observed. Analysis gave 39 per cent iron in ferrous condition. A further 45 minutes roast was made, and resulting product showed 48·2 per cent ferrous iron and a magnetic test showed 91 per cent of the roasted material to be magnetic. The samples all checked very closely, the third sample being roasted for two hours gave results similar to the 1½ hour roast. Analysis of roasted material was:—

	Per cent
Iron (total)	50·00
Copper	1·90
Lead	0·92
Total sulphur	33·30

Of the iron, 97·2 per cent was present in the ferrous condition. A small amount of SO_3 was present as originally in the raw ore. From further analytical data obtained the composition of the roast would be somewhat as follows:—

Fe_8S_9, 79·35 per cent Cu_2S 2·38 per cent FeS_2 2·67 per cent.

Leaching.—A quantity of the roasted material ground to pass a 200 mesh screen (·0029″ opening) was leached with a solution of ferrous and ferric chlorides under conditions of temperatures and agitation similar to those pertaining in the leaching of pyrrhotite.

The reduction of the ferric chloride and leaching did not occur quite as rapidly as with natural pyrrhotite, 210 grams ore requiring 7 hours leaching, with a solution containing 308 grams Fe as ferrous chloride and 209 grams Fe as ferric chloride, excess of ore considering copper and lead, being about 6·5 per cent. However, results showed extraction of 85 per cent of the iron, 42 per cent of the copper, and practically 100 per cent of the lead.

GENERAL CONCLUSIONS

The results obtained in these tests, though being of a very preliminary nature, show that this method of treatment for pyrrhotite and pyrite is very promising. So far as the iron is concerned it would appear that recoveries of over 80 per cent could readily be made.

Concerning copper, the results are not so encouraging, the copper going partly into solution and partly with the gangue. This problem, as well as the separation and recovery of other metallic sulphides occuring in these ores, will be gone into in the investigation contemplated.

The laboratory at present is not equipped with electrolytic apparatus, and so it has not been possible to study the electro-deposition phase of the process. However, plans are being prepared with this end in view, and it is hoped that a continuous circuit suited to the proper study of this process will be installed in the near future, when the process may be investigated under conditions approaching more closely those that would obtain in actual practice.

Test No. 183

THE LOW GRADE ORE FROM THE MURRAY MINE

H. C. Mabee

This test was made on a low grade sample of ore from the Murray Mine, British America Nickel Corporation, Nickelton, Ont., and was conducted with the object of determining if the precious metal values were concentrating with the nickel and copper in the process of flotation, or in which of the products they were reporting. Attention was not paid so much to obtaining the best flotation results, as this had been done in former tests on the ore (See Summary Report of the Mines Branch 1921, Test No. 160), but to the getting of sufficient of the different flotation products for assaying.

To obtain this a number of flotation tests were run, using in all 13 kilos of the ore, and the different products combined.

The assay results of these products, as well as the percentages in the products, appears below.

Product	Weight grams	Assays						Value of precious metal content
		Copper per cent	Nickel per cent	Gold oz./ton	Platin. oz./ton	Palladm. oz./ton	Rho. Irid., etc. oz./ton	
							*	1
Concentrate.....	1,635	2·90	6·40	0·027	0·032	0·043	0·005	$7 40
Middling.....	1,418	0·27	1·65	0·013	0·016	0·015	0·003	3 54
Tailing.....	9,947	0·05	0·21	0·0009	0·0011	0·0026	.....	0 30
Feed.....	13,000	0·43	1·14	0·0055	0·0067	0·0009	0·0009	$1 55

*Results are no doubt low due to oxidation and volatilization in fuzing and scorifying.
[1] Figured from December, 1922, quotations on these metals.

Product	Weight grams	Percentages in Products						Value of precious metal
		Copper	Nickel	Gold	Platin.	Palladm.	Rho., Irid., etc.	
Concentrate.....	12·6	84·4	70·3	61·8	60·9	60·0	.....	60·3
Middling.....	10·9	6·8	15·7	25·7	26·4	18·0	.....	24·9
Tailing.....	76·5	8·8	14·0	12·5	12·7	22·0	.....	14·8
Feed.....	100·0	100·0	100·0	100·0	100·0	100·0	.....	100·0

On this grade of ore the test shows that 60 per cent of the precious metal values are recovered in the concentrate, 25 per cent in the middling, and 15 per cent remains in the tailing. The middling product representing 10·9 per cent of the original feed would be reground and refloated, producing a total recovery of 75 per cent of the precious metal values from this grade of ore.

The result of this test demonstrates fairly conclusively that the gold and the platinum group metals are following the nickel-copper concentrate in the flotation process.

Test No. 184

THE PRECIOUS METAL VALUES IN THE NICKEL-COPPER PYRRHOTITE ORES OF THE SUDBURY DISTRICT. THE LOW GRADE ORE FROM No. 3 (FROOD) MINE

H. C. Mabee

This test was conducted on a low grade sample of ore from Mine No. 3 (Frood) of the International Nickel Co., Copper Cliff, Ont., with the object of determining if the precious metal values were reporting with the nickel and copper in the flotation concentrates, or in what flotation product they were reporting. Attention was not paid so much to obtaining the best flotation results, as this was demonstrated by former tests on this ore (See Summary Report of the Mines Branch, 1921, Test No. 155), but to obtaining sufficient quantities of the flotation products for accurate determination of the precious metal contents.

A sample consisting of 13 kilos of ore was crushed and flotation tests run on the whole in 1 kilo lots, and the products combined. The results of this test are contained in the table following:—

Product	Weight grams	Assay						Precious metal values content
		Nickel per cent	Copper per cent	Gold oz./ton	Platin. oz./ton	Palladm. oz./ton	Rhod., Irid., etc. oz./ton	
Concentrate.....	2,050	5·88	7·55	0·036	0·062	0·072	0·006*	$13 18
Middling.....	1,490	0·93	0·20	0·012	0·019	0·022	0·004	4 47
Tailing.....	9,460	0·16	0·05	0·002	0·0027	0·0038	trace	0 62
Feed.....	13,000	1·15	1·25	0·008	0·014	0·017	0·0014	$3 04

*Results are no doubt low due to oxidation and volatilization in fusing and scorifying.

Product	Weight grams	Percentages in Products						Precious metal values content
		Nickel	Copper	Gold oz./ton	Platin. oz./ton	Palladm. oz./ton	Rhod., Irid., etc. oz./ton	
Concentrate.....	15·8	80·6	95·5	66·8	70·3	68·5	.....	68·4
Middling.....	11·5	9·3	1·8	16·1	15·7	16·3	.....	16·8
Tailing.....	72·7	10·1	2·9	17·1	14·0	15·2	.....	14·8
Feed.....	100·0	100·0	100·0	100·0	100·0	100·0	.....	100·0

The above results on this particular grade of ore show that 68 per cent of the total precious metal values are recovered in the concentrate, 17 per cent appears in the middling, and 15 per cent is retained in the tailing. In this case again, 11·4 per cent of the original feed represented in the middling would be reground and floated, producing a total recovery of 80 per cent of the precious metal values in the process of concentration by flotation.

IV

OTHER TEST WORK OF THE DIVISION

W. B. Timm

Crushing tests were made by R. K. Carnochan on 533 pounds of Nepean sandstone.

Several shipments of clay, totalling 2,540 pounds, were ground and prepared for the use of the Public Schools of Ottawa.

A shipment of supposed gold ore was submitted by Mr. Geo. Cheney. This was found to contain no values in gold and silver, so no further test work was performed.

A sample of rock from Mr. H. Stuart, Dauphin, Man., was identified as greenstone of no commercial value.

A sample, identified as hematite, was submitted by Mr. Dalmore Constantineau of Ottawa.

A supposed gold ore from Mr. Wm. Allen, N.B., was found to contain no gold or silver.

Some test work was performed on a shipment of feldspar by Mr. Norman Davis in the laboratories.

V

ADDITIONAL EQUIPMENT FOR THE LABORATORIES

W. B. Timm

A Callow pneumatic flotation unit, consisting of two rougher cells, and two cleaner cells, capacity 5-10 tons in 24 hours, was installed, to conduct the large scale or tonnage check tests on samples of over two tons. With this unit a mill flow sheet can be followed, closely approaching actual practice, and the results of the small scale laboratory tests verified. The capacity of this unit conforms to that of the grinding capacity of the 4′ 6″ Hardinge ball mill when grinding to 100 mesh or finer.

The unit is so arranged that the feed can be split going to both roughers, each followed by a cleaner cell, or followed by two cleaners, one after the other, or the entire feed can go to one rougher and the tailings to the other, the concentrate from both passing on to separate cleaners, or first one cleaner followed by another cleaner. This arrangement gives almost any desired combination.

A Heavi-Duty type of electric muffle furnace was installed for assay work. This furnace with its special cast alloy tray was found to be most satisfactory for assay work. It is much easier to manipulate and gives a more accurate control of temperatures than the oil and gas furnaces formerly used for this work.

A large electric drying oven was built and installed in the sample room for the drying of products from the small scale test work. The oven is giving the utmost satisfaction for this work.

FUELS AND FUEL TESTING DIVISION

I

CARBONIZATION OF PEAT IN COMMERCIAL HARDWOOD DISTILLATION OVENS

R. E. Gilmore and Harold Kohl

In August, 1922, commercial size carbonization experiments were made on a car lot of air-dried-machine peat. These tests were carried out in co-operation with the Standard Chemical Company of Canada in their plant situated at Longford, Ontario. The ovens used were of standard size and the carbonization process was the same as that used in the regular hardwood distillation practice. Every convenience and much appreciated help was freely given by the officials and operators of the Standard Chemical Company, with whom, under instructions from B. F. Haanel, Chief of the Division of Fuels and Fuel Testing, the senior author had made previous arrangements. The junior author was present and supervised the actual carrying out of the experimental runs.

The purpose of these experiments was two fold, first, to determine the feasibility of carbonizing Canadian air dried peat to obtain charcoal and other chemicals by the same destructive distillation process as used for hardwoods; and second, to find out to what extent peat may be considered as a substitute for hardwood in the manufacture of charcoal and chemical products. The scope of the tests were, therefore, limited to the carbonization process as used in the hardwood distillation industry, which may be termed low temperature destructive distillation in which the temperature is controlled in the neighbourhood of 300° C. (572° F.).

Products obtainable from peat, other than charcoal, are peat tar oils, pyroligneous liquor containing ammonium acetate, methyl (alcohol) spirits and gas. The aim was, as with hardwood, to obtain as large a yield as possible of liquid products, viz., methyl spirits and light oils. The charcoal yield, as a result of this procedure, is also high compared with other carbonization processes and is of a good quality. The yield of gas and ammonia obtainable will necessarily be comparatively low.

Results of carbonization experiments carried out on a large laboratory scale by the above authors are on hand and will be used for comparative purposes in this report. The results published by the Fuel Research Board—England—on "The Carbonization of Peat in Vertical Gas Retorts," viz., Technical Paper No. 4, will also be referred to. Besides briefly reviewing the salient features of the carbonization process used commercially for hardwoods a comparative summary of the results on experiments, first, by low temperature carbonization in the laboratory; second, by low temperature carbonization in commercial hardwood distillation ovens; and third, by high temperature carbonization in vertical gas retorts, will be included in this report.

DESCRIPTION OF OVEN AND CARBONIZATION PROCESS

In Figure 13, a sketch of the oven is given, which oven is of the standard rectangular steel plate construction. At the front end is a full size door, and at the rear a 30-inch gas exit opening. The dimensions of this type of oven are roughly 6 feet wide by 8 feet high by 26 feet long. The door, which is of cast iron, is by means of steel keys and asbestos lined grooves made practically gas tight against its frame which in turn is riveted to the oven proper. The bottom of the oven is fitted with steel rails, on which three iron buggies to hold the fuel are run into and out of the oven. The exit pipe links up with a tubular condenser set in a vertical position immediately behind the oven. From the chamber, above the goose neck liquor discharge at the bottom of the condenser, two pipes are provided for the uncondensible gas, one to the gas main to the boiler room and the other to the fire chamber under the front of the oven.

The steel oven is hung from the roof of the firebrick setting—two ovens to one setting—with heating flues on all surfaces except the front. Below and at the front of the oven is the fire chamber for burning soft coal or sawmill waste. Immediately in front of the oven is a turn table to handle the buggies, which is used both in charging and discharging the iron buggies. The top of the firebrick and tile oven settings is used as a drying floor for acetate of lime.

The heating flues are so arranged that the flames and their hot gases travel from the fire box backward along the bottom of the oven and also upward along the sides and back of the oven. The flames do not play directly on the steel oven, but by means of holes suitably arranged in a firebrick archway, properly constructed lengthwise under the whole length of the oven, these hot flames are uniformly distributed. After passing backward along the bottom and upward along the walls of the oven the passage of the flue heating gases is forward over the top of the oven before returning through horizontal flues under the acetate drying floor to the chimney exits at the rear.

The capacity of an oven, such as was used in these peat carbonization experiments, is 6 cords, i.e. of hardwood, equal roughly to 12 tons. As a ton of the air dried peat containing 25 per cent moisture occupies practically the same space as a ton of hardwood of equal moisture content, the capacity of an oven is roughly 12 tons of peat, four tons to each of the three buggies.

The fuel to be carbonized is charged into a hot oven immediately after the previous charge of charcoal has been withdrawn. The duration of the carbonizing period extends from 24 to 30 hours between charging and discharging. The buggies of hot charcoal are withdrawn from the oven into an air tight cooling chamber, where it remains for a time equal to that of the succeeding carbonizing period. After being further cooled for a second period in a second cooling chamber similar to the first, the cooled charcoal is allowed to remain on the buggies in the open sheds for another 24 hours before bagging and loading on railway cars.

In the regular carbonization operation, as soon as the wood is charged into the oven, hot from the previous run, a quick hot fire is made under the oven to start what is known as the primary destructive distillation. The first distillate which commences running within an hour or so after charging, is composed mostly of water and is accompanied with very little if any gas. In about nine hours, tar is noticed in the distillate and the flow of gas increases rapidly. This is known as the "breaking point"

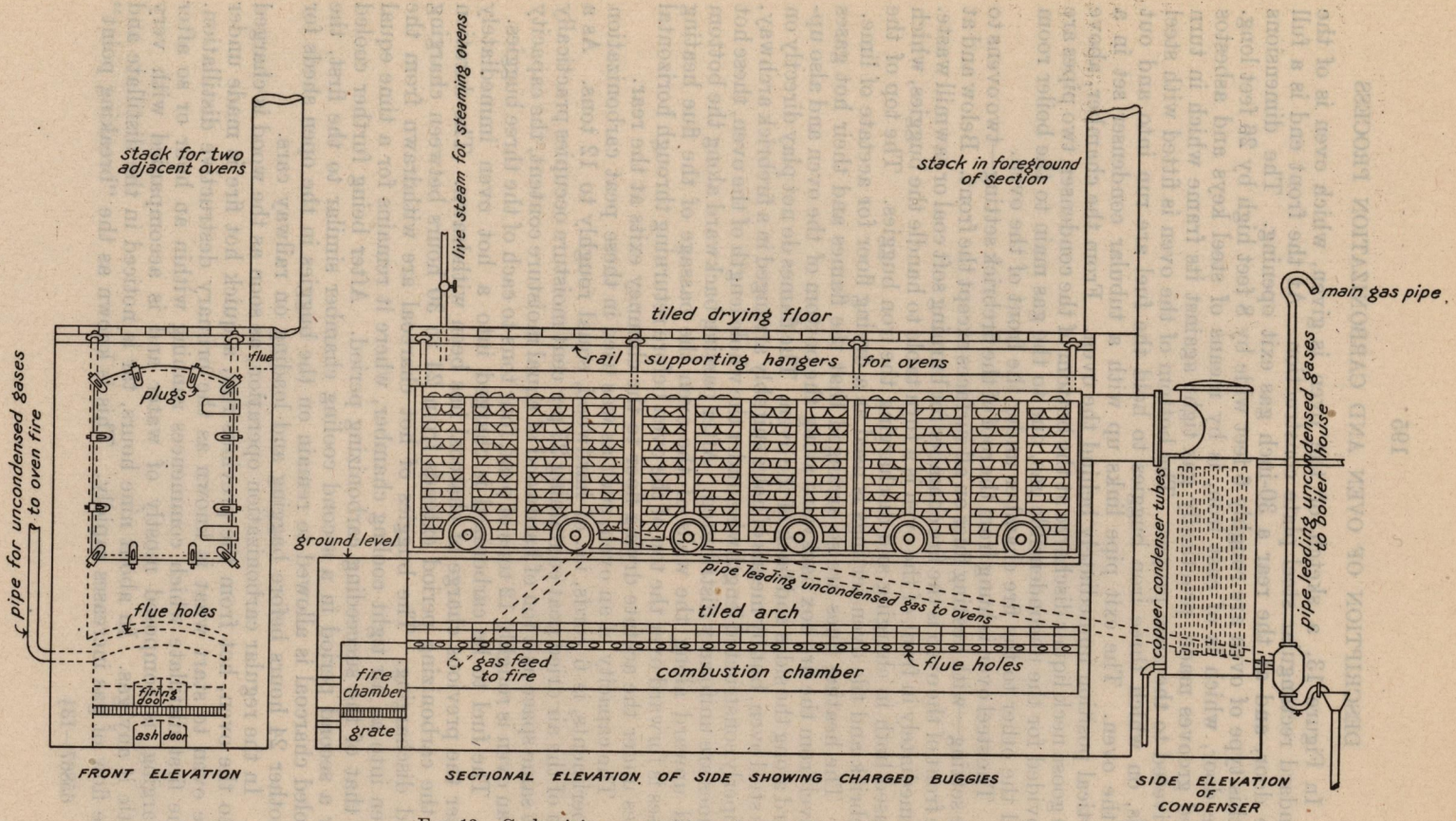


FIG. 13. Carbonizing oven with condenser. Capacity 12 tons of peat.

and is an indication of an exothermic reaction within the oven. At this point or before, depending on the condition of the fuel charged, in respect to age and moisture content, and on the comparative rates of flow of both liquor and gas, the fire under the oven is checked and regulated so that the destructive distillation reaction within the oven is controlled at the proper rate to insure maximum yields of liquid products. By means of recording pyrometers inserted in the gas exit pipe midway between the back of the oven and the condenser, the rate of the reaction is followed and recorded. These exit gas temperature charts are carefully watched and made use of by the fireman. At a certain time the gas given off is allowed to enter and to be burned under the oven, where it either displaces or supplements other fuel necessary to carry out and complete the carbonization. The rate of flow of liquor increases to a maximum at about the sixteenth hour, and is maintained at this rate until the reaction is nearly completed. The last two to three hours are used for completing the carbonization and for sweeping the gases out of the oven with steam before opening the door for discharging the buggies of hot charcoal.

For fuller description of the carbonization process as used for hardwoods, the literature[1] on the subject should be consulted.

The crude products resulting from the destructive distillation of hardwoods, other than charcoal, are tar oils, pyroligneous acid, and gas. The pyroligneous acid is the only raw material from the ovens that requires further treatment to obtain marketable products. By a process of chemical treatment and rectification this pyroligneous acid aqueous distillate is refined to produce (*a*) crude 95 per cent vol. wood alcohol, containing methyl alcohol, acetone, etc., (*b*) light naphtha and creosote oils, and (*c*) acetic acid in the form of grey acetate of lime. Similar products to these, with the addition of ammonia, is to be expected from peat.

The reasons for, and the merits of using this low temperature carbonization process for peat are as follows:—(*a*) The yield of methyl alcohol and allied products and also of the acetate products will be higher by this process than by carbonization processes generally used for coal. (*b*) At temperatures used, namely between 300° and 350° C., as high, if not higher yields of tar oils of high quality are obtainable as with any other low temperature process. (*c*) The peat charcoal, though still high in volatile matter, closely resembles hardwood charcoal, for which it was hoped peat charcoal would be a substitute.

Experimental

The carload of air dried peat used in these tests was supplied by the Peat Committee from the bog at Alfred, Ontario, the analysis of which was as follows:—

	As received basis	Dry basis
Proximate analysis—		
Moisture, per cent	25·6	
Ash, per cent	4·6	6·2
Volatile matter, per cent	48·5	65·3
Fixed carbon, per cent	21·3	28·5
Sulphur, per cent	0·25	0·33
Nitrogen, per cent		1·5
Calorific value—		
B.Th.U., per pound	7040	9470

[1]See bibliography at end of Bulletin No. 74, Forestry Branch, Department of the Interior, Ottawa, viz.,—"Distillation of Hardwoods in Canada."

This peat consisted of irregular lumps varying in size from 2 to 3 inches in diameter to pieces as large as an ordinary brick. The process of manufacture at the bog consisted of macerating the wet peat containing about 90 per cent water, and then after spreading on the ground and cutting into the required sizes allowing it to dry in the open air. The condition of the air dried peat on arrival at Longford was not as good as that sold as household fuel. It was loaded into the buggies from a pile on the ground by means of a steam shovel, and in this way differed from hardwood, which must be loaded by hand.

Two experimental runs were made on August 7 to 10 inclusive. In the first run three buggies containing slightly over four tons each were charged into the oven, and for a second each of the three buggies used contained about three and one-third tons. The data sheets for the runs are as follows:—

Run No. 1

Weight of peat charged—25,868 lbs ..... 12·9 tons
Equivalent of dry peat—19,360 lbs ..... 9·7 tons
Time and date charged into oven ..... 8 a.m., August 7
Time and date drawn from oven ..... 7 a.m., August 9
Duration of run, 36 hours.

Flow and Composition of Oven Liquor

Time in hours	Temperature of exit gas °F.	Temperature of exit gas °C.	Gals. per hour	Total liquor	Wet tar	Aqueous	Acidity as acetic	Remarks
				gals.	p.c.	p.c.		
0	100	38						
2	170	77	6	12				
4	270	132	22	56				
6	360	182	29	114	0·5	99·5	0·40	
8	420	216	46	217	10·0	90·0	0·96	Condenser hot.
10	420	216	51	320	1·0	99·0	0·54	" "
12	420	216	44	408	0·0	100·0	0·72	" cold.
14	460	238	35	478	0·5	99·5	0·81	" "
16	500	260	38	555	5·0	95·0	0·84	" hot ½ down.
18	510	265	41	638	5·0	95·0	0·96	" " ½ "
20	510	265	41	720	7·0	93·0	0·78	" " ½ "
22	510	265	34	788	8·0	92·0	0·78	" " ½ "
24	500	260	30	848	12·0	88·0	0·70	" " ½ "
26	515	265	29	906	28·0	72·0	0·70	" " ½ "
28	535	270	28	963	31·0	69·0	0·60	" " ½ "
30	560	295	28	1,014	31·0	69·0	alkaline	" " ½ "
32	580	310	30	1,081	40·0	60·0	"	" " ½ "
34	580	310	18	1,128	43·0	57·0	"	" " ½ "
36	580	310	13	1,144	45·0	55·0	"	" " ½ "

Average temperature of exit gases, 560°F. (293°C.).

Fuel used under the oven, 1 load of wood waste—sawdust and wood chips—(i.e. other than peat gas) equivalent to about 1¼ tons of soft coal.

Yield of peat charcoal, 9,728 lbs. (15 p.c. Breeze).

Total oven liquor measured 1,144 imp. gallons.
Tar oils 208 "
Aqueous (liquor) 936 "

Methyl (alcohol) spirits in liquor— 1·10 c.c. per 100 c.c.

Ammonia (NH_3) in liquor — 0·45 grams per 100 c.c.

Run No. 2

Weight of peat charged—20,408 lbs..... 10·2 tons.
Equivalent to dry peat—15,183 lbs..... 7·6 tons
Time and date charged into oven..... 8 a.m., August 9
Time and date drawn from oven..... 2 p.m., August 10
Duration of run, 30 hours.

Flow and Composition of Oven Liquor

Time in hours	Temperature of exit gas		Gallons per hour	Total liquor	Wet tar	Aqueous liquor	Acidity as acetic	Alcohol (methyl)
	°F.	°C.		gals.	p.c.	p.c.	p.c.	p.c. vol.
0.....	270	130						
2.....	275	135	7					
4.....	320	160	8	29				
6.....	375	190	36	102		100	0·2	
8.....	420	215	34	169		100	0·5	0·5
10.....	465	240	46	262	5	95	0·6	1·6
12.....	500	260	52	366	5	95	0·6	1·6
14.....	520	270	42	451	10	90	0·6	1·6
16.....	540	280	38	508	8	82	0·9	1·7
18.....	560	290	35	583	11	89	0·6	2·7
20.....	560	290	42	653	13	87	0·6	2·3
22.....	560	290	33	719	6	94	0·6	2·3
24.....	570	295	17	747	16	84	0·6	2·2
26.....	570	295	16	790	20	80	0·4	2·0
28.....	580	300	14	823	25	75	alkaline	2·6
30.....	600	310	8	852	30	70	"	3·0

Average temperature of exit gases, 570° F. (309° C.).
Fuel used under the oven, 2,654 lbs. of slack coal (soft).
Yield of peat charcoal, 7,868 lbs. (15 p.c. Breeze).
Total oven liquor as measured, 852 imp. gallons.
Tar oils, 124 "
Aqueous liquor, 728 "
Methyl (alcohol) spirits in liquor, 1·30 c.c. per 100 c.c.
Ammonia (NH_3) in liquor, 0·45 grams per 100 c.c.

Operation remarks.—In the first run the buggies used were loaded nearly to the top of the side bars. The same buggies loaded to this level with hardwood cordwood held two cords each, equivalent to roughly 2 tons per cord. It was, therefore, noticed that peat of 25 per cent moisture content had practically the same bulk as hardwood of equal moisture content.

In experiments previously carried out on peat in the laboratory it had been necessary to allow the condenser water to warm up, to prevent the thick tar oils from clogging the (spiral) condenser tube. At the same time it was necessary to keep the water, especially at the bottom of the condenser, cool enough to condense the low boiling alcohol constituents. These were kept in mind, but despite precautions a little difficulty was experienced during the first half of run No. 1. In the latter part of this run, however, and in run No. 2, the flow of oven liquor was fairly uniform. In regular practice, using peat, a combination of a warm condenser followed by a cold condenser or scrubber would overcome any difficulty of condensing both the thick tar and the low boiling liquids.

Care was taken to look for indications of exothermic reaction. With hardwoods with moisture contents of about 25 per cent and lower, as previously remarked, more or less sudden increases of liquor and gas are noticed at about the seventh to the ninth hour, and if the fire under the oven is not checked and controlled at this point or previous the temperature within the oven rises too high and the consequent rate of reaction increases to such an extent that serious loss in valuable liquid products is more than likely to ensue. As, however, the moisture content of the hardwoods ranges higher than say 25 per cent the exothermic reaction is not so noticeable. For the high moisture hardwoods constant external heat is required to carry on and complete the reaction. During the two experimental runs with peat no appreciable exothermic reaction was noticeable. Whether this was due to too high a moisture content or was normal to peat of this moisture content is difficult to say. The shape of the curves in the constituents of the gas—*see* Diagram IX—was, however, the same as those on a typical run using hardwood.

The comparative quantities of tar oils and aqueous distillate in the oven liquor were observed from samples taken in a graduated glass cylinder every hour. After standing, a more or less complete separation took place when readings were taken. The total oven liquor was pumped to a tank and measured. After settling, measurements of the total separated clean liquor and settled tar respectively were made.

A flowsheet of the process is given as Fig. 14, and Fig. 15 is the temperature curves on the recording pyrometer chart for run No. 2. The crude factory operations follow closely those used for hardwoods, but the laboratory refining of the crude peat products is slightly different from that for hardwood.

Examination of Products

The peat charcoal produced was passed over a square, $\frac{3}{8}$ inch mesh screen, and what passed through was termed breeze. The percentage of screened lumps and breeze was roughly 85 and 15 respectively. Bag samples of each were taken for analysis. Barrel size samples of both the aqueous distillate and settled tar were taken to the laboratory in Ottawa for analyses and reserve samples. The aqueous distillate, which may be termed peat pyroligneous acid, was a clear straw colour, and the tar oils were of a consistency that flowed readily when poured from one vessel to another.

The pyroligneous acid from peat differed from that from hardwood in that the former contained an appreciable amount of ammonia and amine compounds. On the other hand, the acid content of the peat liquor was small as compared to that from hardwood. In ordinary hardwood pyroligneous acid practice the liquor is neutralized with lime before distilling off the crude alcohol. The residue on evaporation would yield brown acetate of lime.

For the recovery of the alcohol product from the peat liquor two different methods are feasible. When the settled liquor is over-neutralized with lime and distilled, similar to the practice for liquor from hardwoods, the ammonia compounds are liberated and come over with the alcohol vapours. By this method nauseous vapours are given off which are troublesome to work with in the laboratory. For this reason, and also for the reason that for the recovery of the alcohol free from amine compounds, the peat liquor was distilled from acid reaction.

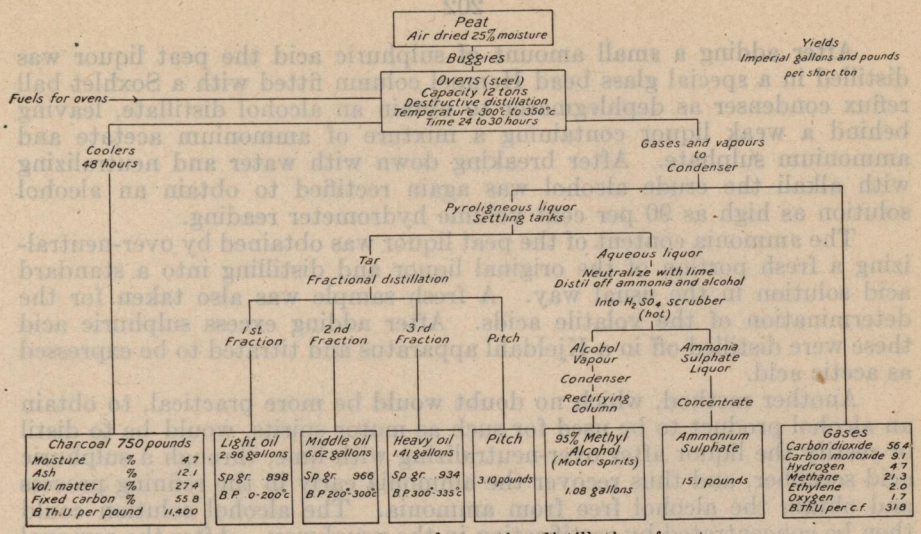


FIG. 14. Flow sheet, destructive distillation of peat.

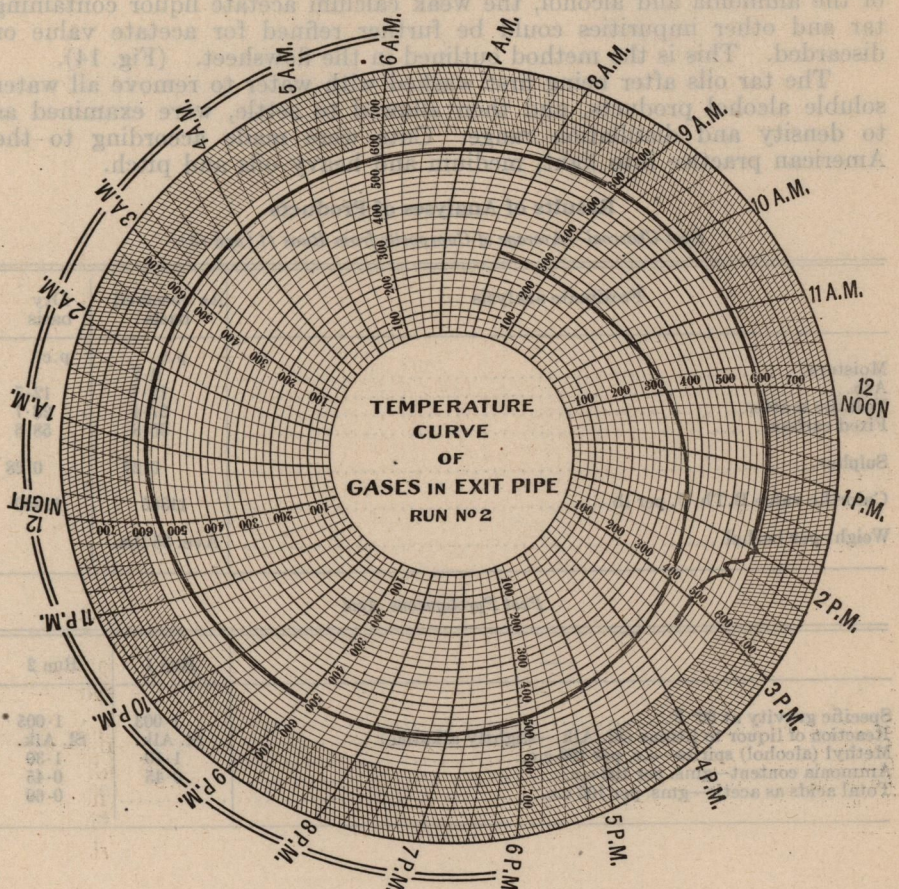


FIG. 15. Temperature curve of gases in exit pipe.

After adding a small amount of sulphuric acid the peat liquor was distilled in a special glass bead Hempel column fitted with a Soxhlet ball reflux condenser as dephlegmator to obtain an alcohol distillate, leaving behind a weak liquor containing a mixture of ammonium acetate and ammonium sulphate. After breaking down with water and neutralizing with alkali the crude alcohol was again rectified to obtain an alcohol solution as high as 90 per cent volume hydrometer reading.

The ammonia content of the peat liquor was obtained by over-neutralizing a fresh portion of the original liquor and distilling into a standard acid solution in the usual way. A fresh sample was also taken for the determination of the volatile acids. After adding excess sulphuric acid these were distilled off in a Kjeldahl apparatus and titrated to be expressed as acetic acid.

Another method, which no doubt would be more practical, to obtain an alcohol product to be used for such as motor spirits, would be to distil or rectify the liquor after over-neutralizing with lime, through a sulphuric acid scrubber, and thus recover the ammonia early in the refining process and obtain the alcohol free from ammonia. The alcohol solution could then be concentrated by rectification in the usual way. After the removal of the ammonia and alcohol, the weak calcium acetate liquor containing tar and other impurities could be further refined for acetate value or discarded. This is the method outlined in the flowsheet. (Fig. 14).

The tar oils after being first washed with water to remove all water soluble alcohol products, and then allowed to settle, were examined as to density and distillation range. Cuts were made according to the American practice into light, medium and heavy oils and pitch.

Results of Analyses of Products

Peat Charcoal—Average of Composite from Runs (1) and (2)

Proximate analysis	As received basis	Dry basis
	p.c.	p.c.
Moisture	4·7	
Ash	12·1	12·7
Volatile matter	27·4	28·7
Fixed carbon	55·8	58·6
Sulphur	0·27	0·28
Calorific value B.Th.U. per lb	11385	11940
Weight per bushel	Ave. 35 lbs.	

Peat Pyroligneous Acid

—	Run 1	Run 2
Specific gravity at 60° F	1·003	1·005
Reaction of liquor to litmus (Sl. Alk.—slightly alkaline)	Sl. Alk.	Sl. Alk.
Methyl (alcohol) spirits—c.c. per 100 c.c	1·10	1·30
Ammonia content—gms. per 100 c.c	0·45	0·45
Total acids as acetic—gms. per 100 c.c	.....	0·60

Peat Tar Oils—Composite Sample Runs 1 and 2

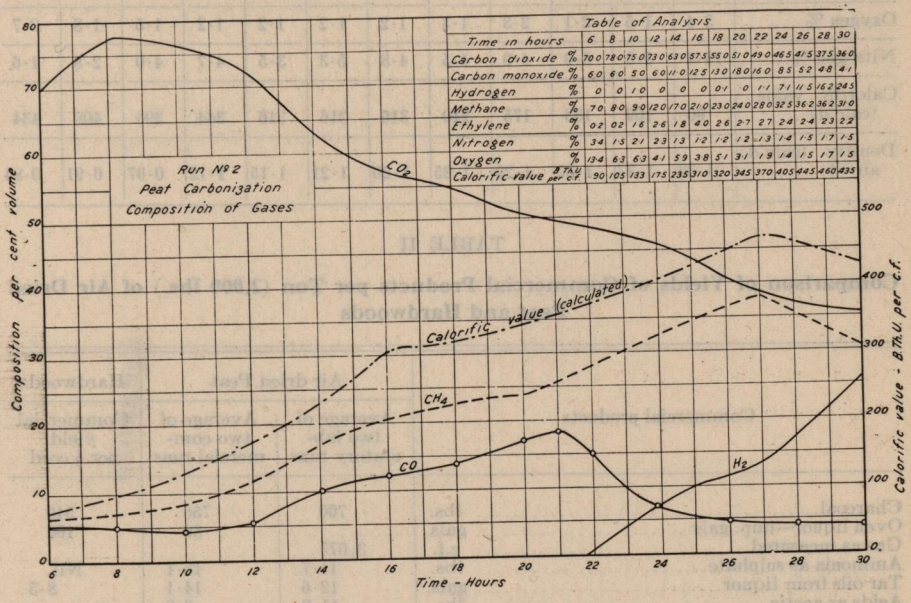
Specific gravity at 60° F. of the crude oil—0·968

Distillation results	Per cent volume		Sp. gr. of fraction
	Wet	Dry	
Water	3·0		
Light oil up to 200° C	18·0	18·5	0·898
Middle oil 200 to 300° C	47·0	48·5	0·966
Heavy oil 300 to 335° C	10·0	10·3	0·934
Pitch (by diff.) above 335° C	22·0	22·7	

Peat Gases—by Low Temperature Carbonization

The yield of oven gas during the commercial scale run was not obtained. Some idea of this yield is, however, available from the laboratory scale runs referred to in the next paragraph. During the runs at Longford, samples of the uncondensed gas were taken at intervals of two hours. Table No. I gives the analyses of these samples taken during No. 2 run and Diagram IX shows the results of these analyses in curves where percentage composition is plotted against time. The figures in the table inserted in Diagram IX were taken from the rounded off curves.

Diagram IX



As referred to in the early part of this report laboratory scale tests had previously been carried out. These tests were on 12 to 13 pound samples of air dried peat averaging 12·5 per cent moisture and under

conditions similar to laboratory efficiency experiments on different hardwoods. Two runs only on peat were made, and while they were not entirely satisfactory, the results are of interest here in comparison with those obtained on the commercial scale. In Table II the laboratory results have been recalculated for air dried peat on a 25 per cent moisture basis. For the sake of comparison, or rather contrast, the average yield per ton of hardwoods (4,000 lbs. per cord) has been added.

TABLE I

Composition of the Gases—Run No. 2

Samples taken at intervals of 2 hours

Sample No.	1	2	3	4	5	6	7	8	9	10	11	12
Time in hours	6th	8th	10th	12th	14th	16th	18th	21st	23rd	25th	27th	30th
Carbon dioxide %	70·1	78·0	75·2	74·0	62·2	56·6	57·0	48·8	49·0	44·2	37·8	36·4
Carbon monoxide %	6·2	6·3	4·0	4·2	11·8	12·1	12·4	19·5	10·6	6·0	5·5	4·1
Hydrogen%	.....	.....	1·1	.....	.....	.....	0·1	.....	.....	10·4	13·0	24·5
Methane %	6·7	8·9	9·1	12·0	18·4	21·2	23·4	24·2	31·6	31·2	37·7	30·5
Ethylene %	0·2	.....	1·6	2·6	1·8	4·1	2·6	2·5	2·9	2·7	2·4	2·2
Oxygen %	3·4	1·5	2·1	2·3	1·3	1·2	1·2	1·2	1·2	1·5	1·8	0·7
Nitrogen %	13·4	5·3	6·9	4·9	4·5	4·8	5·3	3·5	4·7	4·0	2·8	1·6
Calorific value (calc'd)	90	109	133	175	240	316	316	316	344	396	408	434
Density (calc'd) air=1	1·32	1·37	1·32	1·29	1·25	1·20	1·21	1·15	1·12	0·97	0·91	0·84

TABLE II

Comparison of Yields of Commercial Products per Ton (2,000 lbs.) of Air Dried Peat and Hardwoods

Commercial products		Air dried Peat		Hardwoods
		Average of two laboratory runs	Average of two commercial runs	Commercial yield per ½ cord
Charcoal	lbs.	700	756	540
Oven liquor—imp. gals.	gals.	.....	86	100
Gas as measured	c.f.	3,675		
Ammonia as sulphate	lbs.	11·7	15·1	Nil
Tar oils from liquor	gals.	12·6	14·1	8·5
Acids as acetic	lbs.	11·8	6·8	
80 per cent acetate of lime	"	19·5	8·3	105
Alcohols as methyl—				
Imp. gals. 100 p.c. vol. basis		1·07	1·03	4·00
Imp. gals. 95 per cent vol. basis		1·12	1·08	4·25

Attempts to obtain a reliable weight balance were not successful due to lack of facilities necessary for measuring the gas and also to loss of charcoal, both by burning while transferring from oven to coolers and to fines in handling. A rough balance sheet, totalling runs 1 and 2, may be drawn up as follows:—

Total—25 p.c. moisture—air dried peat charged	47,267 lbs.	
Charcoal as weighed 17,596 lbs.	37·2%	of peat charged
Oven liquor including tar 19,960 lbs.	42·2	" "
Gas—from laboratory estimate	13·5	" "
Unaccounted for loss	7·1	" "
	100·0%	

As will be noticed the figure of 1·000 has been used as the density of the oven liquor, and the gas yield used is the same as that determined by laboratory runs, the results of which on other products checked the commercial run fairly closely.

Technical paper No. 4, Fuel Research Board, entitled "The Carbonization of Peat in Vertical Gas Retorts," has also been referred to above. The air dried peat used in these commercial size experimental runs, on the 25 per cent moisture basis as received from the bog, conformed to an average analysis as follows:—3·3 per cent ash, 46·6 per cent volatile matter, and 25·1 per cent fixed carbon with a calorific value of 7215 B.Th.U. per lb. This analysis corresponds closely with the air dried peat used for the low temperature runs at Longford, with the exception of ash, viz.: 3·3 as compared with 4·6. A comparison of yields and composition of the commercial products as given in Table III should be of interest here.

TABLE III

Comparison of Commercial Products in Hardwood Oven and in Vertical Gas Retorts

Yields given on a basis of short ton (viz., 2,000 lbs.) of peat charged

Products	Low temperature carbonization in wood ovens	High temperature carbonization in gas retorts
Peat charcoal		
Per cent of air dried peat charged	37·2	25·1
Pounds per ton	74·5	50·3
Per cent ash (dry basis)	12·7	9·9
Per cent volatile matter (dry basis)	28·7	3·9
Calorific value B.Th.U. per lb	11940	12650
Peat tar oils—imp. gals. per ton		
Total produced	14·1 gals.	14·2 gals.
Specific gravity at 60° F	0·968	0·992
Fraction to 170° C. (dry basis)	6·7%wt	2·5% wt
" 170 " 230° C	25·8	13·9
" 230 " 270° C	17·5	15·4
" 270 " 335° C	24·0	34·7
Pitch and loss (by difference)	26·0	33·5
Aqueous liquor products per ton		
Total liquor (imp.) gallons	86	87
Ammonium sulphate, lbs.	15·1	22·6
Crude alcohol (imp.) gallons	1·08	0·34
Gas—c.f. per ton	less than 4,000	about 12,000
Crude motor spirits, (imp.) gallons		
Alcohol—plus oils below 170° C	2·10	0·75

Value of Peat Carbonization Products

Peat charcoal is to be considered the main product of low temperature carbonization with the tar oils, gas and alcohol as by-products and of importance in the order given. By the high temperature processes especially where a large yield of gas is desired, the charcoal may be an equal or secondary product as compared with the gas. The charcoal produced by the wood ovens in many respects closely resembles wood charcoal. Its weight per bushel, however, is heavier, averaging 35 lbs. as compared with 21 pounds for hardwood charcoal. At these weights per bushel, a ton of air dried peat will yield only 21 bushels as compared with 26 bushels from a ton (half a cord) of hardwood. In view of the sale of the hardwood product in paper bags as domestic fuel, peat charcoal to be sold by bulk in a similar way is of less value per unit weight. The value of the peat charcoal from a ton of air dried peat is, therefore, disappointing in comparison with wood charcoal from an equal weight of hardwood. As a substitute for hardwood charcoal to be handled and sold in bags, at the same price per bag or bushel, 35 pounds of peat charcoal will net no more than 21 pounds of hardwood product.

Special uses of peat charcoal may be considered the same as hardwood charcoal, such as starting and maintaining short lived fires as in kitchen ranges, train dining room stoves, and in locomotive engines. For metallurgical purposes peat charcoal, or coke as it is sometimes termed, may prove adaptable. Should its structure be such that it could be used for this purpose its low sulphur content should be in its favour.

Peat charcoal as a household fuel, other than having much higher heating value, has not many superior merits to air dried peat, say with moisture content 20 per cent or below. The charcoal besides being dirty and difficult to handle in bulk tends to burn too fast in a furnace. As an open grate fuel, however, it may be considered a substitute for cannel coal, and should be worth in small or large sacks as much as cannel coal which sells as high as $1 per 100 pound lots. Discounting this figure, say 25 per cent, we may arrive at a conservative (wholesale) value of peat charcoal of $\frac{3}{4}$ cents per pound, or $15 per short ton.

The peat tar oils on preliminary examination appear to be an oil mixture containing both creosotes and parafin, resembling wood tar oils on one hand and crude petroleum on the other. A certain amount of light oils may be recovered as crude motor spirits, and other valuable products may be there, but for the time being it would be unwise to consider the heavier oils other than as crude fuel oil, say at a value of 10 cents per gallon. Peat tar oils are good flotation reagents[1], and for this purpose may be worth as high as 20 to 25 cents per gallon as crude unrefined oils.

The alcohol yield from peat was not as high as had been expected. The highest strength obtained so far in the laboratory was 90 per cent volume as per alcoholmeter reading. The alcohol obtained in these commercial runs has a similar boiling point range to that of crude wood alcohol, and is miscible with water. As a motor fuel the 95 per cent product should be of value, especially when blended with certain of the

[1] Crude peat tar oils as such and fractions from the same have been experimented with in the laboratories and plant of the Ore Dressing and Metallurgical Division of the Mines Branch, and found to be quite satisfactory in the treatment of many Canadian ores by the flotation process.

light tar oils. This methyl spirits light peat oil product in the semi-refined state should be worth at least 20 cents per gallon for motor fuel purposes. Another possible use for these methyl spirits is as denaturant for grain alcohol for the denatured alcohol market.

The cost of obtaining the different products from the peat liquor in comparison with their value, is a point to be considered. In respect to alcohol it is questionable if its value would be appreciably above the cost, as its percentage in the aqueous liquor is near the 1 per cent figure, at which point the cost of refining is said to approach the value as a concentrated (crude) product. In respect to recovering the acid value of the liquor as acetate of lime the cost is considered prohibitive. For this reason no credit should be claimed for an acetate by-product.

The gas produced by carbonizing peat by the hardwood distillation process would be burned under the ovens, thus allowing no credit from this product. The ammonia content of the uncondensed gases was not determined.

The ammonium sulphate yield given is that from the oven liquor to be obtained by the method outlined in the flow sheet. Its value may be placed at say 2 cents per pound.

The estimated value of the products from 2,000 pounds of air dried peat may be summed up as follows:—

Peat charcoal, 745 lbs. at ¾ cents per lb	$5 60
Heavy tar oils, 13 imp. gals. at 10c. per gal	1 30
Motor spirits, 2 imp. gals. at 20c. per gal	0 40
Ammonium sulphate, 15 lbs. at 2c. per lb	0 30
Total value of products per short ton	$7 60

A rough estimate of the value of the products of a cord (4,000 lbs.) of hardwood at the crude factory, using recently prevailing wholesale prices, is as follows:—

Hardwood charcoal, 1,100 lbs. at 1c. per lb	$11 00
Crude alcohol, 8½ imp. gals. at 50c. per gal	4 25
Acetate of lime (80%), 210 lbs. at 2c. per lb	4 20
Tar oils, 17½ gals. at 10c. per gal	1 75
Total value of crude products per cord (2 tons)	$21 20

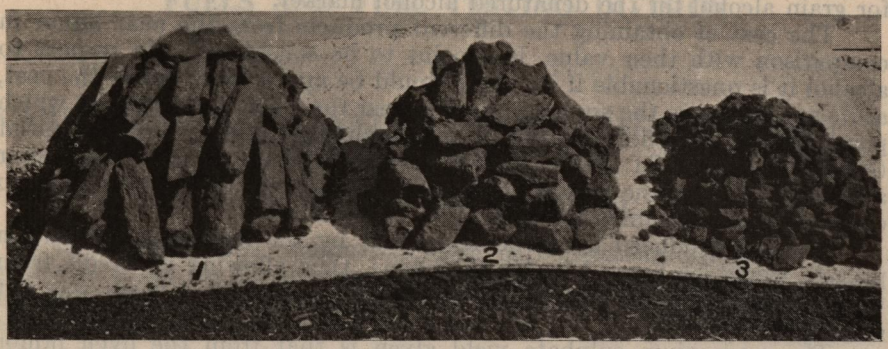
Using these figures for the value of charcoal and other products from peat which would likely vary considerably from time to time, similar to the market value changes for hardwood products, and noting that a ton of hardwood is equivalent to only half a cord, the following comparison may be made:—

Value of products per ton of hardwoods	$10 60
Value of products per ton of peat	7 60

It will be noticed that the ratio of the value of charcoal to the total value of products from peat would, according to the above figures, be roughly 75 per cent as compared with about 50 per cent for hardwoods.

Further work is being carried out on the examination of both the alcohol product and the tar oils from these commercial size low temperature runs on air dried peat, the results of which will be published in future reports of investigations.

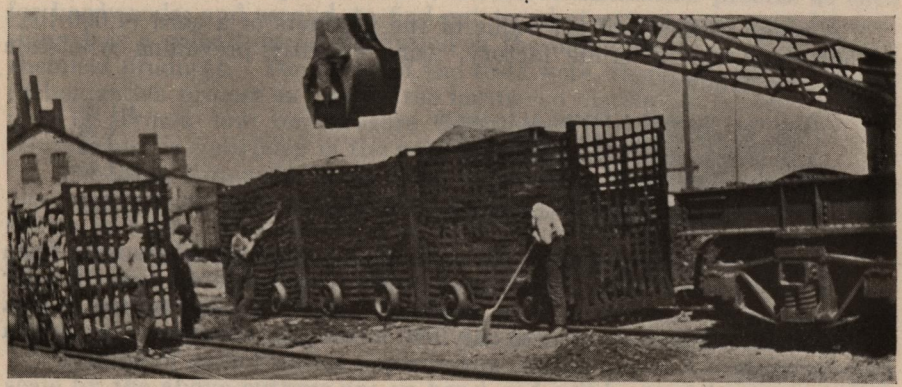
PLATE IV.



Air dried peat and peat charcoal

1. Large size (machine made) briquettes; 2. Peat—broken pieces—as carbonized; 3. Peat charcoal produced

PLATE V.



Loading the peat into the buggies by steam shovel

SUMMARY AND REMARKS

In this investigation a car lot of air dried peat with slightly over 25 per cent moisture content was carbonized in plant size hardwood distillation ovens, under low temperature conditions. The rate of reaction was controlled at about 300° C., and the yields of the commercial products were measured, a preliminary examination of which has been made.

Air dried peat bricks when handled and carbonized in the same manner as hardwoods responded to the same method of oven firing, but required slightly more fuel to carry on the carbonization during the later half of the run, which extra fuel was supplied by the oven gases produced. Although the oven firing did not require special attention to avoid a too rapid increase of the internal temperature of the ovens, there were indications of exothermic reaction, but not to the same extent as with hardwood. The moisture in the peat, however, apparently masked any appreciable outward signs of such reaction. The commercial products obtained were peat charcoal, peat tar oils, crude alcohol, and ammonia, of value in the order given. The alcohol when blended with certain light oils may be considered as a crude motor spirits product.

Peat charcoal in many respects resembles hardwood charcoal, and as a special fuel for quick, hot and short lived fires may be considered a substitute. The quality of the alcohol from peat, the yield of which is disappointingly low in comparison with hardwoods, has not yet been thoroughly examined.

The weight per bushel of peat charcoal averaged 35 pounds as compared with 21 pounds per bushel for hardwood charcoal, and the yield per ton of raw material carbonized was roughly 21 bushels from peat and 26 bushels from hardwood. At the same price per bushel to be marketed in bulk similar to the hardwood product, the value of peat charcoal is, therefore, only about 0·8 of the value of the charcoal from equal weight of hardwood.

The lumps of peat charcoal produced by carbonizing air dried peat were irregular in shape and not as jet black as wood charcoal. Although apparently more friable than charcoal from hardwood, the handling properties of the peat charcoal are such that it can be used as a fuel in the condition produced and does not need to be briquetted.

The value of the products from air dried peat is estimated as three-quarters of the value of the products from an equal weight of hardwood and for peat the value of the charcoal as a percentage of the value of the total products is much higher than for hardwood.

When once the cost of air dried peat in large quantities is known, the yield figures given in this report may be used for determining the economic feasibility of carbonizing peat similar to the low temperature process as used for hardwood. Until such a time, however, as the production of air dried peat more than meets the demand as a household fuel, it is doubtful if the price of the raw material will allow the carbonization of peat to develop into an industry to either supplement a successful peat harvesting enterprise or in conjunction with the already established hardwood distillation industry.

65867—14

II

REPORT ON THE TREATMENT OF OIL SHALE FROM NEW BRUNSWICK BY THE RYAN OIL DIGESTION PROCESS

A. A. Swinnerton

In order to ascertain the merits of a new process for the treatment of oil shales the so called Ryan oil digestion process was examined. This was in accordance with the plan of investigational work on Canadian oil shales provided for in the Mines Branch investigations for 1921. A demonstration of this Ryan process was given in New York by Mr. Wm. H. Hampton in the presence of Mr. B. F. Haanel, Chief of the Fuel Testing Division.

The experimental retort as used to demonstrate this process was duplicated, and further experiments on oil shale from New Brunswick were made by the author in the Fuel Testing Laboratory during 1922. Examination of the patent drawings for larger continuous operating scale apparatus has also been made. The purpose of this paper is, therefore, to report on the technical points of the process, keeping in mind the application of the same to a commercial scale, according to the claims and drawings of patentee.

The process, according to U.S. Patent 1,327,572, January 6, 1920, Henry D. Ryan, Colorado, consists of the "digestion of the shale in a closed retort with a body of relatively heavy oil under the influence of

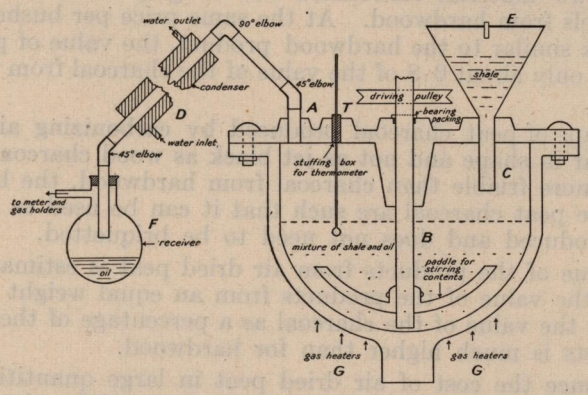


FIG. 16. Diagram of laboratory Scale Digester for oil shale.

heat," and "at a temperature insufficiently high to effect substantial distillation of heavy fractions, but high enough to effectively liquefy heavy hydrocarbons in the said solids." The claim is made that this treatment results in the conversion of the bitumen in the shale from its relatively solid state into a liquid, and this without the evolution of any considerable quantity of fixed gases, presumably at the expense of the heavier oil fractions.

The apparatus used in the laboratory experiments reported herein consisted of a cast iron retort or pot ten inches internal diameter and eight inches deep, having a flange, so that by means of gaskets, bolts and nuts the cover could be made tight in order to prevent gas and oil leaks. As seen in Fig. 16 this cover was fitted with the following: an offtake A leading to a condenser D; a thermometer receptacle T, shaft B carrying a paddle for agitating the contents, and a brass charging funnel C. The retort was heated by gas, using a large special high power burner. Besides the retort and burner the apparatus train consisted of a 36-inch Liebig condenser, fitting into an adapter which in turn was fitted to a large flask F. From a second opening in the top of the flask, a continuation of the closed system consisted of rubber and glass tube connections through a gas meter to a gas holding tank. In the condenser and separating collecting flask respectively ice cold water and ice was used to insure condensing maximum oil vapours.

The procedure as witnessed in the experimental run made in New York, and as duplicated in the Fuel Testing Laboratory experiments, was as follows: 3,500 grams, i.e. approximately $7\frac{3}{4}$ pounds of heavy lubricating oil were poured into the retort and the top bolted on. The retort was then heated by a gas heater to a temperature of 680–700° F., and 1,800 grams or more, i.e. about 4 pounds of finely divided shale was slowly introduced through the funnel C, Fig. 16, care being taken to prevent the escape of vapours by always having the funnel full of shale. When all the shale had been introduced the cover E was screwed on and heating continued for an hour.

During the reaction the contents were kept stirred by means of the paddle, which was driven by an electric motor (not shown). The small amount of vapours obtained by heating the heavier oil and the vapours given off during the reaction escaped by the offtake pipe A, and after passing through the condenser D, collected in the receiver F.

The retort was then allowed to cool down to room temperature, when the cover was removed and the contents transferred to a basket centrifuge, kerosene being added to thin down the mixture of shale and heavy oil and to wash out the digester.

The mixture of shale, heavy oil and kerosene was then centrifuged, further kerosene being added if necessary to wash the last traces of the heavy oil from the shale residue. The liquid obtained from the centrifuge was collected and the dry residue in the basket transferred to a retort and distilled, the oil driven off being added to that already obtained. The light oils were distilled off and the heavy oil remaining could be used for digestion of more oil shale.

The following is the summary of the results of the demonstration run made in, and reported from New York:—

	lbs.
Oil charged in still	7·00
Stripping to 680° F	0·045
Net oil	6·955
Uncondensed vapours during reaction	0·105
	6·850

	lbs.
Digestion oil recovered by distillation	6·390
Apparent shortage	0·460
Total diluting and wash oils used	12·500
Vaporizing, atomizing transfer and leakage losses	0·210
	12·290
Uncondensed vapour during distillation	0·065
	12·225
Distillation recovery	11·420
Oil with tailings	0·805
Oil recovered from tailings by heating to 750° F	0·890
Wash and diluting oils charged to tailings	0·805
Oil gain	0·085
Oil and water from distilling tailings to 920° F	0·225
Oil and water from shale reaction 680-690° F	0·775
	1·065
Less water from both sources	0·055
Oil	1·010
Less apparent digestion oil shortage	0·460
Net oil gain	0·550
Shale charged	4·00

This yield of 0·550 pounds per 4·00 pounds shale is equivalent to 38·63 U.S. gallons per ton. Gravity (38° Be equivalent 0·83 sp. gr.).

The composition of the shale used was given as follows:—

	per cent
Water	1·375
Oil	12·250
Gas (by difference)	2·500
Fixed carbon	4·375
Ash	79·500
	100·00

The quantity of oil obtained from 4 pounds shale by destructive distillation was reported as 8·75 fl. ozs. gravity 29° Be equivalent to 34·12 U.S. gallons per ton. So that according to these figures the net gain by the Ryan process was 38·63 minus 34·12 U.S. gallons per ton, equivalent to 3·76 imp. gallons.

In the above figures, the net oil gain of 0·550 lbs. includes the losses resulting from the digestion and from centrifuging, viz. 0·105 and L·210 pounds respectively. This method of figuring the oil gain is apparently based on the assumption that the losses may be accounted for as recoverable light oil vapours. In the experimental runs made as a check on the above run, there were losses first as uncondensable gas during the digestion and second while washing the residue with kerosene in the centrifuge.

Accordingly the net gain of oil due to shale oil should not include these losses. The following summary of the (New York) run would seem to be more proper:—

Shale used		4·00 lbs.
Oil used—		
Digestion oil		7·00 "
Wash oil		12·500 "
Total		19·500
Oil obtained—		
Stripping digestion oil		0·045 "
Distillate from shale		0·755 "
Oils from centrifuge		17·875 "
Oils from tailings by distillation		1·115 "
		19·790 "
Less water		0·055 "
Total		19·735 "
Yield—		
Oil obtained	19·735 lbs.	
Oil used	19·500 "	
Gain	0·235 "	

This gain of 0·235 pounds would be the net gain due to oil extracted from the oil shale and is equivalent to a yield of 17 U.S. gallons per ton, which is approximately only half of that obtainable by destructive distillation.

Experimental

In the apparatus described above, of which the retort as shown in Fig. 16 was the main part, several experimental runs were made for the purpose, first, of checking the results already reported from the demonstration run, and second, to determine what happened at each stage of the operation.

The oil used for digestion was Veedol heavy, which on analysis gave the following results:—

Specific gravity at 60° F	0·896
Flash point (closed test. P.M.)	410° F. (210° C.)
Fire point (open)	464° F. (240° C.)
Viscosity (Redwood) at 100° F	332 secs.
210° F	51 secs.
Evaporation test—	
400° F. for 6 hours	Loss 3·9 p.c.

Several mechanical difficulties were encountered in operating the retort. A mechanical screw feed for the shale was tried but had to be abandoned as the finely divided shale would not feed properly, so feeding by hand was resorted to. The brass funnel was kept full of shale and pushed in with a rod, and when the weighed amount of shale had been charged and the funnel was empty, the lid was screwed down. The 200 mesh woven wire filter cloth supplied with the centrifuge was found to be too coarse to retain the shale residue. It was therefore reinforced with sheets of filter paper as no finer mesh screen could at the time be procured, and this combination gave quite satisfactory results.

The following table gives the details of various runs made, Nos. 1, 2, and 3 were runs made in this laboratory and No. 4 is a recapitulation, for comparison, of the results obtained at New York, reduced to grams.

—	No. 1	No. 2	No. 3	No. 4
Weight of shale used (grams)	1,850	1,812	1,812	1,816
Weight of oil used (grams)—				
Digestion oil	2,860	2,575	2,949	3,178
Wash oils	1,987	1,233	1,931	5,680
	4,847	3,808	4,880	8,858
Oil obtained (grams)—				
Stripping digestion oil	205	.....	.....	20
During reaction	567	225	646	343
From centrifuging	3,753	3,270	3,844	8,118
From tailings by distillation	480	530	519	506
	5,005	4,025	5,009	8,987
Less water	8	10	7	25
Net oil obtained (grams)	4,997	4,015	5,002	8,962
Total oil added	4,847	3,808	4,880	8,858
Oil gained from shale by digestion process (grams)	150	207	122	104
Oil from same amount of shale by destructive distillation in lead bath temperature control app. (grams)	245	239	239	222

In the above runs, difficulty was experienced in accounting for the losses, and attempts to obtain a reliable weight balance were unsuccessful, due mainly to loss of light oil vapours, both from creeping past the retort gasket during digestion and during the washing with kerosene in the centrifuge. The impracticability of trying to carry out this process as per laboratory scale directions was at this stage quite evident. It will be noticed that the largest yield of shale oil was obtained in run No. 2, where the amount of digestion oil was the smallest, and vice versa.

There was in each case a certain amount of uncondensed gases which were measured, but the unaccounted for losses which varied for the different runs ruined all attempts to determine just how much of these losses should be assigned to recoverable oil vapours. Until such a time therefore as the process can be carried out in a closed system similar to the patent drawings as shown in Fig. 17, thus necessitating doing away with open centrifuging, it is concluded that no satisfactory results from such laboratory experiments will be available.

In order however to determine at what stage or stages the losses occurred a further run was made with the following results:—

Input—	
Weight shale treated	1,800 grams
Weight digestion oil	3,039 "
Weight wash oil used	1,383 "
Total	6,222 "
Recovery—	
Distillate	154 grams
Oil shale	6,006 "
Gas	49 "
Total	6,216 "

Loss so far in the process, 6 grams.

The shale and oil mixture was thinned with kerosene oil and centrifuged, the yields being as follows:—

Input—	
Oil and shale	6,006 grams
Wash oil	2,075 "
Total	8,081 "
Recovery—	
Oil	5,827 grams
Shale	1,981 "
Total	7,808 "
Loss during centrifuging	273 grams
Total oil vapour loss	279 "
Total gas and oil loss	328 "

The digestion temperature maintained in these experiments was as per directions from 690 to 700° F. and the shale used was a standard laboratory sample No. 1517 from Albert Mine, the maximum yield from which averaged 13·6 per cent weight by destructive distillation equivalent to slightly over 30 imp. gallons per 2,000 pounds ton.

As a result of a study of temperature conditions in previous oil shale investigational work, and also during the above runs, indications were that the digestion temperature was not high enough to insure the breaking down of the so called kerogen oil forming compounds of the New Brunswick shale used. The shale, after digesting at different temperatures, was extracted in Soxhlet thimbles with carbon disulphide and then submitted to destructive distillation. In all digestions below say 750° F. (400° C.) it was found that varying quantities of shale oil still remained in the shale in a form not readily extracted by kerosene or other low boiling solvents. In the neighbourhood of 750° F. or above, the oil forming constituents of the shale were found to be practically all digested. This is in accordance with the minimum temperature required for the recovery of maximum oil yields from shales by destructive distillation. *See* Mines Branch Investigations in 1921, page 243.

For the New Brunswick shales the digestion must be carried out at temperatures not less than about 750° F. (400° C.). This is evidently not necessary for oil shales from certain other regions. For example, the Californian shales are reported to be different in that the oil is in a readily digestible form and they yield their oil when digested at comparatively lower temperatures[1]. Once however digestion at temperatures as high as 750° C. is necessary, trouble is encountered by the cracking of the heavy digestion oil into lighter oil fractions, with the formation of permanent gases. When this reaction takes place the claims of the process according to the reading of the patent are not substantiated as "the principal object of the invention is the obviation of such losses."

For the purpose of determining the nature of the light oil distillate or strippings during the digestion at temperatures sufficiently high to break down the kerogen oil forming compounds, a series of runs were made on the heavy Veedol oil, first alone, next with sand, then with spent shale, i.e. shale from which the oil had been extracted, and finally with

[1]U. S. Geological Survey Bull. 729, p. 14.

fresh oil shale, all under practically the same conditions. The temperature during the digestion was carried slowly up to a maximum of 750° F. In each case a light yellow acrid distillate along with uncondensable gas was given off and collected. Following are the comparative results obtained:—

—	Veedol oil alone	With sand	With spent shale	With oil shale
Veedol oil used (grams)	1,082	943	1,901	3,039
Other material used "	none	732	500	1,800
Maximum temperature °Fahr.	750	750	750	750
c.f. of gas obtained	–	–	0·21	1·64
Light oil distillate obtained (grams)	83	121	156	154
Sp. gr. of " at 60° F.	0·788	0·794	0·782	0·799
Distillation Range of Light Oil Strippings				
1st drop	64° C.	71° C.	68° C.	64° C.
10 p.c. vol.	101	117	106	120
20 "	131	147	128	154
30 "	157	170	147	176
40 "	184	197	166	199
50 "	203	220	188	217
60 "	223	245	206	242
70 "	247	277	225	263
80 "	278	284	250	295
90 "	328	320	288	342

On examining the above distillation ranges and allowing for variations in rate of heating it will be noticed that the light oil distillates from the heavy (lubricating) oil used did not vary appreciably when digested alone or with different finely divided solids. The strippings from Veedol alone correspond quite closely with that from Veedol and oil shale, which shows that when oil shale is digested with a heavy oil such as Veedol lubricating oil, at a sufficiently high temperature to break down the kerogens of the oil shale, there is a partial cracking which reacts (though apparently not as readily) on the heavy digestion oil as well as on the heavier bitumen fractions of shale oil itself.

DISCUSSION OF PATENT DRAWINGS

Fig. 17 is a copy of what is termed "the single sheet of drawings" accompanying the patent application referred to above. Designations of the main parts of the apparatus have been added to the drawing alongside the original numbers.

A brief description of the apparatus is interpreted as follows. The crushed shale is fed into retort 2 heated by fire on grate 15. In the retort it is rotated and digested by means of a revolving screen 5 in heavy oil 14. The light oil vapours or strippings pass up and are condensed in 50. The hot digester oil, carrying with it the extracted shale oil and finely divided shale residue, is dropped into an upward flow of kerosene wash oil in

conical shaped tank 18. The kerosene wash oil laden with the shale oil passes on into tanks 36, 37 and 38, which are fitted with heating coils 39, while the clayey residue drops into chamber 25, fitted with steam jacket 26, from which the dry (supposedly solvent free) residue is discharged at 28. The oil mixture in tanks 36, 37 and 38 after being r'd of the light oils is returned through pump 45 to the digesting retort. The light oils, which will be the main product of the process after being distilled off by means of pressure steam in coils 39, are condensed and cooled in 32 before passing into holding tank 47 to be drawn off at valve 48.

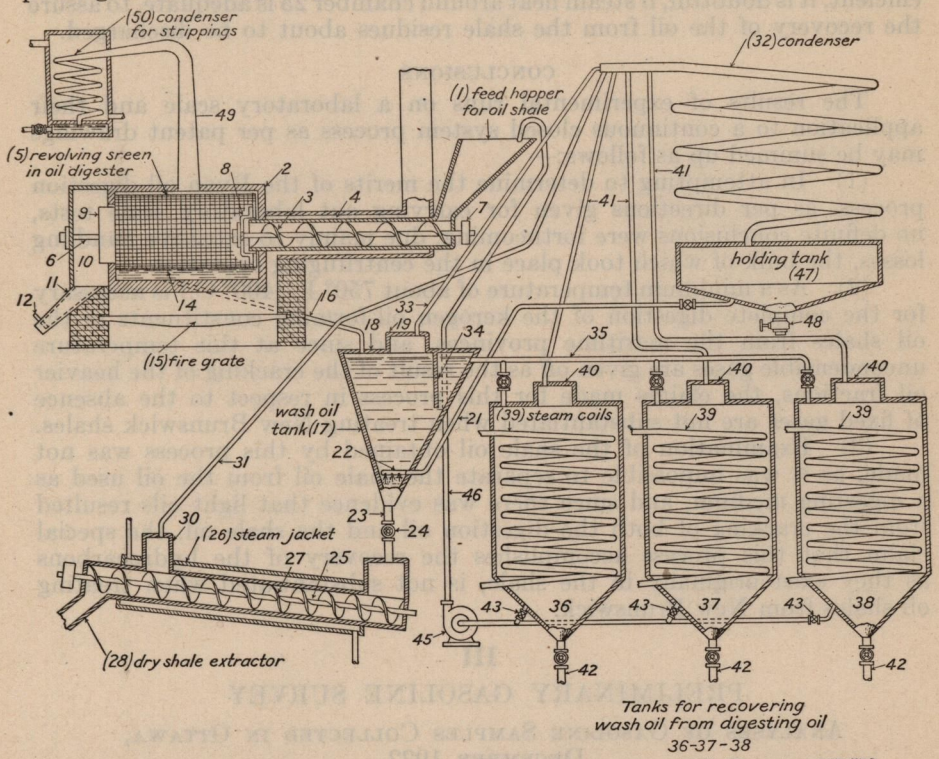


FIG. 17. Copy of patent drawings, Ryan Oil Digestion Process. U.S. Patent, 1,327,572.

The apparatus as shown in the drawing is a closed system, evidently necessary in order to prevent evaporation losses. No provision is made for uncondensible gases, presumably on the assumption that none are formed. The results of the foregoing experiments indicate that this assumption, when applied to oil shales from New Brunswick, is questionable. The process and apparatus were apparently designed to treat oil shales that do not require temperatures as high as 750° F. to liberate the oil from their oil bearing components. For shales that require this temperature or above, to break down the kerogen constituents either in the digestion operation or in a final heat treatment of extracted shale residue, it is doubtful if this oil digestion process has superior merits to an ordinary destructive distillation process.

The heat for distilling off the light wash oils from the heavy digestion oil, in tanks 36, 37 and 38 is evidently that contained in pressure steam, say a maximum of 328° F. (164° C.) at 100 pounds absolute pressure. Unless live or superheated steam, or both, is used this restricts the consistency of the wash oils to that of a light to medium gasoline or crude naphtha and does not allow of kerosene, such as advised for laboratory experiments, being used, as the average initial point of kerosene is above the temperature of steam at ordinary boiler pressures. Again, unless the washing of the heavy digestion oil from the clayey shale residue is highly efficient, it is doubtful, if steam heat around chamber 25 is adequate, to assure the recovery of the oil from the shale residues about to be discharged.

CONCLUSIONS

The results of experimental runs on a laboratory scale and their application to a continuous closed system process as per patent drawings may be summed up as follows:—

(1) In attempting to determine the merits of the Ryan oil digestion process, as per directions given for carrying out laboratory scale tests, no definite conclusions were forthcoming, due mainly to excessive handling losses, the bulk of which took place in the centrifuging operation.

(2) As a minimum temperature of about 750° F. (400° C.) is necessary for the complete digestion of the kerogen oil-forming constituents in the oil shales from the maritime provinces, and since at this temperature uncondensible gases are given off as the result of the cracking of the heavier oil fractions, the claims made for this process in respect to the absence of fixed gases are not substantiated when treating New Brunswick shales.

(3) Examination of the shale oil obtained by this process was not made, as it was impossible to separate the shale oil from the oil used as a digesting medium, and since there was evidence that light oils resulted from the cracking of both the digestion oil and the shale oil, the special claim that this process accomplishes the recovery of the hydrocarbons as they exist originally in the shale, is not substantiated when treating oil shales from New Brunswick.

III

PRELIMINARY GASOLINE SURVEY

Analyses of Gasoline Samples Collected in Ottawa, December 1922

P. V. Rosewarne

INTRODUCTION

At different times there has arisen in various parts of Canada a spasmodic agitation for a standard specification for gasoline. It would seem logical before attempting to set a standard, if such action were deemed advisable, to first find out the characteristics of the gasoline being sold on the open market in different parts of the country.

This report covers the analyses of twenty samples sent to, or collected by the Fuel Testing Laboratory in December, 1922. In the future it is planned to extend this preliminary local survey to a Dominion wide survey to be reported annually in the Mines Branch Summary of Investigations.

To this report has been added, for the benefit of the non-technical reader, a few paragraphs as an appendix to explain the meaning and significance of the different tests. It is hoped that it may help toward a better understanding of a rather complex subject.

ANALYSES OF SAMPLES COLLECTED

Of the twenty samples, five samples were from the wholesale dealers and the remaining fifteen were from the retail distributing stations in the city of Ottawa. In general the analyses of the wholesale and retail samples sold under the same trade name checked quite closely, but there was considerable variation between the gasolines of different grades, and between gasolines of the same grade, for example, Grade II as discussed later.

After determining the specific gravity, corrected to 60 degrees, Fahrenheit, each sample was subjected to a distillation range test and was examined for unsaturated compounds by determining the iodine absorption value. All determinations were made in duplicate and the values given are the average of the two estimations.

The specific gravity was found by means of the hydrometer and the Westphal balance. The distillations were run according to the method recommended by the United States Bureau of Mines[1]. The iodine values were determined by the Hanus method[2], which gave very consistent results.

TABLE I

Results of Analyses

Sample No.	Sp. gr. at 60° F.	Baume degrees at 60° F.	First drop ° F.	20 per cent ° F.	50 per cent ° F.	90 per cent ° F.	End point ° F.	Per cent recovered	Iodine values
2128	0·717	65·3	126	176	221	309	388	.....	2
2129	0·735	60·5	129	197	246	342	423	.....	32
2130	0·734	60·7	126	199	261	339	390	.....	37
2131	0·717	65·3	132	181	225	304	364	.....	1
2132	0·722	63·9	117	174	226	320	400	.....	4
2134	0·719	64·7	133	187	227	316	367	97	4
2135	0·731	61·5	122	194	251	354	423	98	29
2136	0·725	63·1	133	185	228	322	399	98	10
2137	0·736	60·2	123	208	268	342	388	97·5	31
2138	0·736	60·2	127	208	268	342	395	98	31
2139	0·735	60·5	118	205	258	347	388	98	31
2140	0·735	60·5	129	207	270	344	397	97·5	31
2141	0·723	63·6	117	180	229	327	408	98	4
2142	0·718	65·0	135	186	227	306	365	98	1
2143	0·731	61·5	90	141	315	444	478	95	10
2144	0·737	60·0	127	209	268	347	408	98	31
2145	0·724	63·4	90	145	252	430	471	95	5
2146	0·727	62·6	123	183	227	329	405	98	12
2147	0·718	65·0	133	187	227	307	361	98	2
2148	0·717	65·3	135	186	227	306	363	98	2
Average	0·727	62·6	123	187	246	339	399	.....	16

[1] U.S. Bureau of Mines Technical Paper 298. "Methods for Testing Petroleum Products."
[2] Ellis and Meigs. "Gasoline and other Motor Fuel."

DISCUSSION OF RESULTS

The average figures shown above cannot be taken as the actual analysis of the average gasoline sold in Ottawa, because it takes no account of the varying amount handled by the different wholesalers and retailers. It is simply an average of the samples taken. However, it may be considered to be a quite close approximation to the true value and as such it is interesting to compare it with other results obtained in a somewhat similar way and with some published specifications. Diagram X gives in graphic form the average distillation curve for Ottawa as determined above, the average distillation curve for the United States as determined by the Bureau of Mines[1], and a distillation curve representing the specification for gasoline adopted by the Federal Specifications Board of the United States for the use of the various departments and independent establishments of the United States government.

A study of the results shows that these samples can be divided into three distinct grades or groups of gasoline. Grade No. 1 is distributed by two companies[2] under two trade names. Grade No. 2 is distributed by three companies and is retailed to consumers under at least six trade names. Grade No. 3 is distributed by one company and is retailed to consumers under at least two trade names.

It will be noticed that the results of the analysis of some samples in Grade No. 2 come very close in some respects to the results of some samples in Grade No. 1. It was felt, however, that if the limits were enlarged to admit those samples into Grade No. 1, that then samples much more inferior than they are would also come within those limits.

Diagram XI shows graphically the difference in the average results of the distillation ranges of the gasoline in the different groups.

TABLE II

Analyses by Grades

GRADE NO. 1

Laboratory No.	Sp. gr. at 60° F.	Iodine value	First drop ° F.	20 per cent ° F.	50 per cent ° F.	90 per cent ° F.	End point ° F.
2128	0·717	2	126	176	221	309	388
2131	0·717	1	132	181	225	304	364
2134	0·719	4	133	187	227	316	367
2142	0·718	1	135	186	227	306	365
2147	0·718	2	133	187	227	307	361
2148	0·717	2	135	186	227	306	363
Average	0·716	2	132	184	226	308	368

[1] United States Bureau of Mines, Report of Investigations, Serial No. 2444, "Seventh Semi-Annual Motor Gasoline Survey."

[2] In this report the word "company" or "distributor" refers to a wholesale dealer.

DIAGRAM X

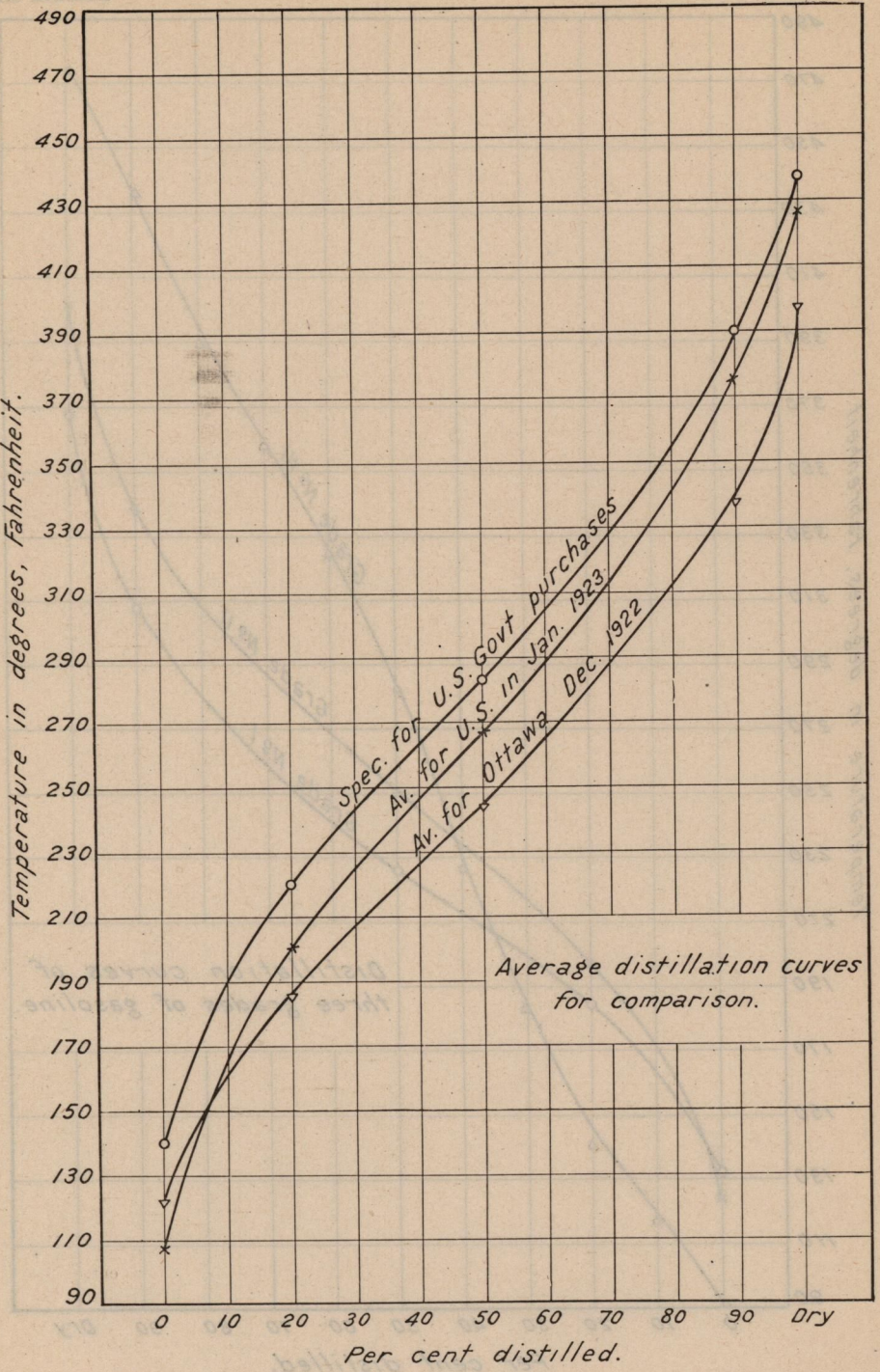

Temperature in degrees, Fahrenheit.
490
470
450
430
410
390
370
350
330
310
290
270
250
230
210
190
170
150
130
110
90
0
10
20
30
40
50
60
70
80
90
Dry
Per cent distilled.
Spec. for U.S. Govt purchases
Av. for U.S. in Jan. 1923.
Av. for Ottawa Dec. 1922
Average distillation curves for comparison.

DIAGRAM XI

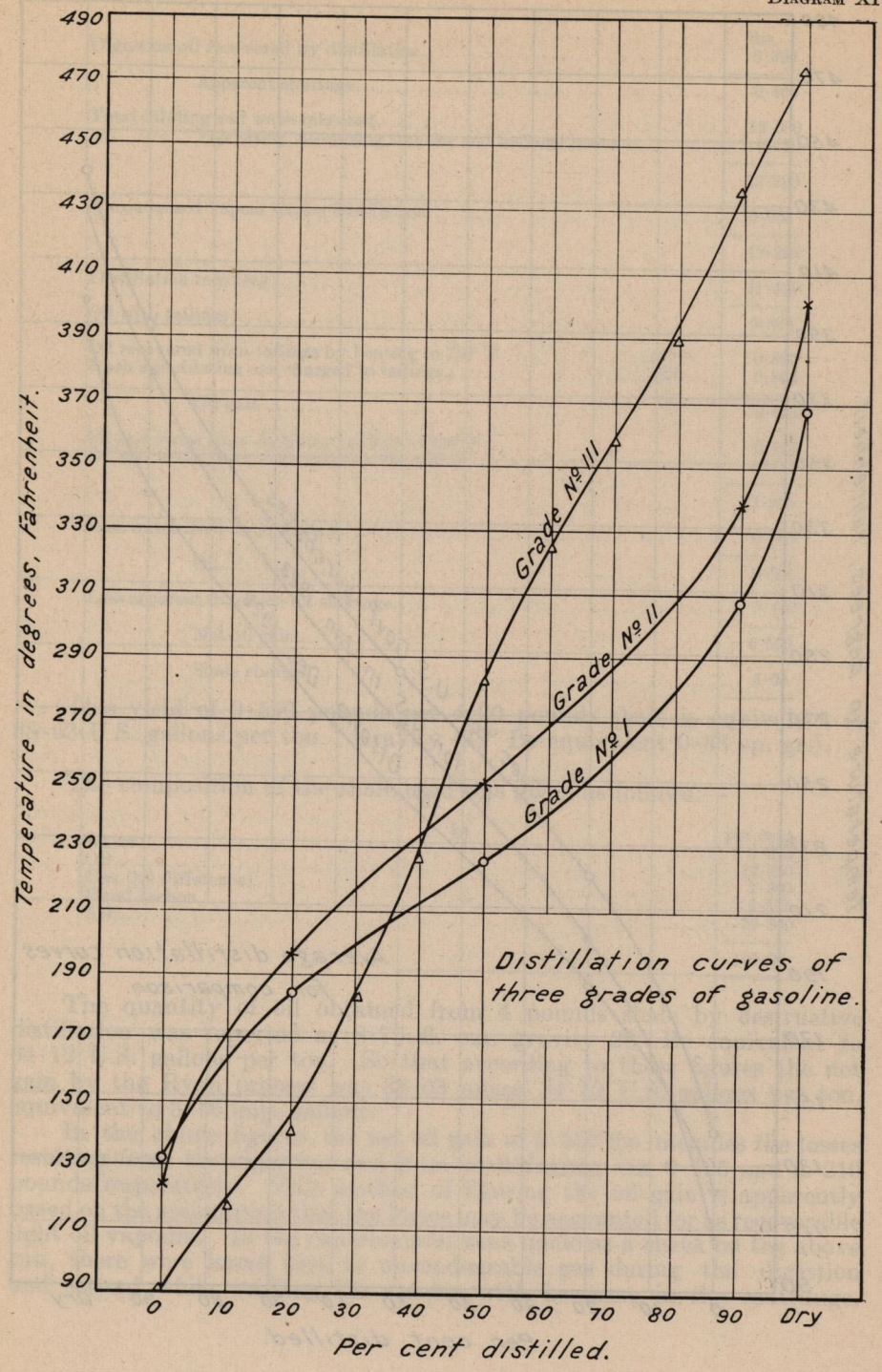

Temperature in degrees, Fahrenheit.
490
470
450
430
410
390
370
350
330
310
290
270
250
230
210
190
170
150
130
110
90
0
10
20
30
40
50
60
70
80
90
Dry
Per cent distilled.
Grade No. III
Grade No. II
Grade No. I
Distillation curves of three grades of gasoline.

TABLE II—*Concluded*

Analyses by Grades—Concluded

GRADE NO. 2

Laboratory No.	Sp. gr. at 60° F.	Iodine value	First drop ° F.	20 per cent ° F.	50 per cent ° F.	90 per cent ° F.	End Point ° F.
2129	0·735	32	129	197	246	342	423
2130	0·734	37	126	199	261	339	390
2132	0·722	4	117	174	226	320	400
2135	0·731	29	122	194	251	354	423
2136	0·725	10	133	185	228	322	399
2137	0·736	31	123	208	268	342	388
2138	0·736	31	127	208	268	342	395
2139	0·735	31	118	205	258	347	388
2140	0·735	31	129	207	270	344	397
2141	0·723	4	117	180	229	327	408
2144	0·737	31	127	209	268	347	408
2146	0·727	12	123	183	227	329	405
Average	0·731	24	124	196	250	338	402
GRADE NO. 3							
2143	0·731	10	90	141	315	444	478
2145	0·724	5	90	145	252	430	471
Average	0·728	8	90	143	284	437	475

TABLE III

Averages for Three Grades

Grade	Sp. gr. at 60° F.	First drop ° F.	20 per cent ° F.	50 per cent ° F.	90 per cent ° F.	End ° F.	Iodine value
No. 1	0·716	132	184	226	308	368	2
No. 2	0·731	124	196	250	338	402	24
No. 3	0·728	90	143	284	437	475	8
Average	0·725	115	174	253	361	415	11

TABLE IV

Summary of Analysis by Grades

—	Grade I		Grade II		Grade III	
	Above	Below	Above	Below	Above	Below
Specific gravity at 60° F	0·715	0·720	0·720	0·740	0·723	0·732
Baume degrees at 60° F	65·8	64·4	64·4	59·2	63·6	61·3
Iodine value	0	7	4	37	5	10
Distillation range:						
Initial point	.....	135° F.	.....	135° F.	89° F.	91° F.
20 per cent by Vol	175° F.	188° F.	180° F.	210° F.	140° F.	146° F.
50 per cent by Vol	220° F.	228° F.	225° F.	270° F.	251° F.	316° F.
90 per cent by Vol	305° F.	317° F.	320° F.	355° F.	429° F.	445° F.
End point	.....	388° F.	.....	425° F.	.....	479° F.

APPENDIX

SOME NOTES ON THE MEANING AND SIGNIFICANCE OF THE DIFFERENT TESTS

Specific Gravity.—The relation between specific gravity and Baume degrees is often imperfectly understood. The specific gravity of a

substance is the weight of a certain volume of the substance divided by the weight of an equal volume of water. It is evident from the above definition that the specific gravity will be high for substances commonly called heavy, such as lead or iron, and will be low for substances commonly called light, such as wood or cork. The same thing applies to liquids. There are heavy liquids, as the sulphuric acid in a storage battery, and light liquids, as ether and gasoline. The two latter liquids are lighter than water and from the definition of specific gravity must accordingly be represented by a fractional or a decimal number less than 1.

The easiest method of comparing the specific gravity of liquids is by means of a hydrometer. A hydrometer is merely a bob or float weighted at one end so that it stands upright in the liquid to be tested. A scale is attached to the upper end. The scale may be divided so that the true specific gravity can be read off directly, or any convenient set of numbers can be used. The Baume scale for light liquids, that is, for liquids lighter than water, is simply a consecutive set of whole numbers, each of which has a definite specific gravity. There was formerly some uncertainty regarding the exact value of the Baume scale and a great deal of confusion arose because different scales gave slightly different readings. In an effort to avoid this confusion the American Petroleum Institute has urged the acceptance of one particular method of calculating the subdivisions of the Baume scale, and this scale is sometimes referred to as the A.P.I. scale.

Practically all liquid petroleum products have a specific gravity less than 1, and the Baume scale for light liquids has been used almost exclusively in the petroleum industry because it avoids the use of fractions and decimals. It is not satisfactory because a high figure on the Baume scale is equivalent to a low specific gravity, and vice versa. Accordingly, many people refer to a high specific gravity or high gravity when in reality a high Baume figure is meant. This accounts for the familiar terms "High Test" and "Low Test" gasoline. "High test" used in this sense means that the Baume figure is high and therefore the specific gravity is low, while "low test" has the opposite meaning. The following table may be used to compare Baume degrees and specific gravities at 60° F.

TABLE V

Specific gravity at 60° F.	Degrees Baume at 60° F.	Specific gravity at 60° F.	Degrees Baume at 60° F.
0·700	70·0	0·755	55·4
0·705	68·6	0·760	54·2
0·710	67·2	0·765	53·0
0·715	65·8	0·770	51·8
0·720	64·4	0·775	60·6
0·725	63·1	0·780	49·5
0·730	61·8	0·785	48·3
0·735	60·5	0·790	47·2
0·740	59·2	0·795	46·1
0·745	57·9	0·800	45·0
0·750	56·7		

Although the specific gravity of gasoline has been used for many years as a test for its quality, at the present time it is not of very much value, because a gasoline may be so mixed or blended as to give any desired gravity within reasonable limits. It may be used as an aid, however, in interpreting the results of the distillation range test.

Distillation Range.—The distillation range is the most important single test used to determine the quality of a gasoline, because it gives the most information regarding the composition of the fuel. A good gasoline should consist of a certain amount of material that boils at a low temperature so that an explosive mixture can be readily obtained. However, a large amount should not be present since then the evaporation loss would be large. A good gasoline should all distil over before too high a temperature is reached or the fuel will not burn satisfactorily. The writer is inclined to believe that the gasoline having the smallest temperature range between the point at which 20 per cent distils over and the point at which 90 per cent distils over is best, providing the initial and end points are within reasonable limits. If the distillation ranges of two or more gasolines are plotted on squared paper, as in Diagram X and Diagram XI, the more desirable sample as far as volatility is concerned has a curve which lies below and is flatter between the above mentioned points than the other. If the curve is very steep, as in Grade III, Diagram XI, the gasoline usually will be unsatisfactory in service because the higher boiling portions will not vaporize and mix properly with the lighter boiling portions so as to form a mixture of gases which will burn evenly and completely in the cylinder of the motor. On the other hand the curve should not be too flat, because then there is a tendency towards knocking and to low mileage due to excessive evaporation.

The amount of evaporation is indicated by the recovery from the distillation. If a very high recovery is obtained, there is likely to be difficulty in starting the cold motor in which the gasoline is being used; if a very low recovery is found, there will be considerable loss through handling, but that defect will be somewhat counterbalanced by greater ease in starting, especially in the winter. It is to be remembered that prevailing temperature has a decided influence on the carburetion of gasoline, and that a fuel that is satisfactory in the summer may not be desirable during the winter months.

In view of the possible exhaustion of the petroleum fields an effort should be made by all users of gasoline to use as low a grade for motor fuel as will operate satisfactorily, and thus make available as large a percentage as possible of the crude petroleum.

Iodine Value.—The iodine value indicates the presence of those chemical compounds which are termed unsaturated. Gasoline which has been obtained by a simple distillation and refining of crude petroleum is composed largely of saturated compounds which give a very low iodine value. When gasoline has been produced by one of the cracking processes a comparatively large amount of unsaturated compounds are present and the iodine value is high. The general tendency has been to condemn unsaturated compounds in gasoline as wholly undesirable, but nevertheless their presence materially increases the available supply of motor fuel without very definitely affecting its combustion. It is reported[1] that up to 12 per cent unsaturates may be present in a gasoline without seriously reducing its value in ordinary use. This is approximately equivalent to an iodine value of 36.

[1] United States Bureau of Mines, Technical Paper 181, "Determination of Unsaturated Hydrocarbons in Gasoline."

CERAMICS AND ROAD MATERIALS DIVISION

Howells Fréchette
Acting Chief of Division

Owing to the lack of staff and the disorganized condition of the division following the transfer of Mr. Joseph Keele to the Geological Survey, very little, aside from routine testing of samples submitted by the public, was attempted in the ceramic laboratory, and almost no field work pertaining to ceramics was done during the year.

Investigations on road materials were conducted by Mr. H. Gauthier and Mr. R. H. Picher. The field work consisted of road material surveys in the Rocky Mountains Park, and in Nova Scotia.

A large number of samples of rock, gravel and other road materials collected in the field were subjected to tests[1] in the laboratory during the winter. A number of crushing tests on granite cylinders of various dimensions were made in connection with the study, by the American Society for Testing Materials, of methods of testing rock to determine *crushing strength.*

The following reports set forth the activities of the various officers in the field and laboratory.

I

CERAMIC MATERIALS

Howells Fréchette

Clays and shales.—During the year many samples of clay and shale were submitted for testing by field officers of the Department of Mines and Provincial Bureaus of Mines, and by individuals and corporations from various sections of the country. In practically all cases these samples were given a complete test for brick making, including the determination of the temperature of softening. In the case of such samples as appeared to warrant it, they were tested for suitability for pottery or for other special uses.

In many cases the samples were very unsatisfactory for testing purposes, being too small in amount or carelessly packed. Even for purposes of prel minary tests a sample should not weigh less than two pounds, and a ten pound sample would be preferable. Clay may be shipped either dry or moist, but care should be exercised to see that the containers are perfectly clean and sufficiently strong to withstand rough handling during transportation. When moist samples are shipped they should be in cloth bags or wrapped in cloth and placed in a sound container. All express or postal charges must be paid in advance by the shipper. Failure to do this is a frequent cause of delay and confusion. A definite statement as to the exact locality from which the sample was obtained should be furnished in all cases.

Magnesite.—A research was commenced on the problem of producing a dense, dead-burned product of high dielectric strength from the magnesite of Argenteuil county, Quebec. Owing to pressure of other work this could not be carried to completion, although considerable progress was made.

Pottery.—Very little was done in this line except to test a few clays on the wheel and to work out a satisfactory casting slip for one of the new commercial potteries using Canadian clay.

[1] An explanation of the laboratory tests for road-building stone and gravel was given in the Summary Report for 1921; pp. 272-275.

II

REPORT ON INVESTIGATION OF ROAD MATERIALS ALONG THE HAWK CREEK-McLEOD MEADOWS SECTION OF THE BANFF-WINDERMERE HIGHWAY, ROCKY MOUNTAINS PARK

H. Gauthier

The investigation of available road materials along the Banff-Windermere highway, in the Rocky Mountains Park, which was undertaken two years ago, was resumed and completed during last summer.

At the end of the 1921 field season there remained 25 miles of the road to be surveyed, that is from Hawk creek to McLeod Meadows camp.

During the summer of 1922 this was completed. Every occurrence of road material lying close to the roadway was examined. In carrying on this investigation 38 samples of gravel were collected for laboratory tests. These tests were made in the Road Materials laboratories of the Mines Branch during the winter. They include granulometric analysis, gravel pebble classification, determination of percentage of clay and silt, together with examination of the shape of the pebbles and of the amount of impurities present. The results of these tests are tabulated in this report. They also contain remarks on the suitability of the gravels examined for use in road construction.

A great deal of information regarding the location and character of the available road materials along the entire mileage of the Banff-Windermere highway is now at the disposal of the park officials. There still remains the problem of economically improving the binding and the bearing power of certain classes of material occurring along the road, in order to secure a satisfactory road surface. This problem will be worked upon in the laboratories.

SOURCES OF SUPPLY OF ROAD MATERIALS

The available material suitable for road surfacing along the Banff-Windermere highway, from Castle, at its northern end, to Hawk creek, and from the Kootenay valley to the Columbia valley at its southern end, includes, as stated in previous reports, three classes of road materials: bedrock, gravel and boulder clay.

The subsoil and the character of the gravel and boulder clay deposits along these two sections of the road vary a great deal from place to place, and in several instances for long distances, clay is the only material occurring.

In the Hawk creek-McLeod Meadows section of the road, which was surveyed last summer, the material is much less variable. Stoneless clay and boulder clay occur, but only for very short distances. The subsoil generally consists of gravel containing a varying amount of loam or clay. Gravel is exposed in the road cuts nearly all the way, and deposits where pits could be opened are found at short intervals and close to the road. Because of its common occurrence, gravel constitutes the main class of road material available in this section.

There are a few bedrock outcrops close to the road, but they are, as a rule, composed of soft shales unsuitable for road work. The only occurrence of bedrock suitable for the production of crushed stone is an important ledge of limestone, which forms a cliff over 25 feet in height, on the west side of the road, at approximately one-fourth of a mile north of Mile 8 creek. The exposure is continuous for several hundred yards and the opportunities for quarrying are good. The limestone is fresh and of fairly good quality. Its value as road metal can be compared with that occurring east of Johnson canyon on the Banff-Lake Louise road. The results of laboratory tests made upon a sample collected there are included in the appendix to this report.

Crushed stone could be produced by using boulders, bouldery gravel, or certain talus and morainic materials.

In the production of crushed stone from such material the crusher can be located at the pit, talus or slide, next the road, and the boulders or stone fragments picked up and put into the crusher with little or no preliminary breaking. Boulders form a cheap and often valuable source of roadstone. The disadvantages of using this form of material for road surfacing arise from its heterogeneous character, which may cause one section of a road to wear more rapidly than another or may give rise to uneven wear in the same section of road. In the area surveyed last summer it was found that in many deposits the stone aggregate was almost entirely composed of limestone, therefore a fairly homogeneous crushed stone product could be obtained from that source of material.

Gravel deposits in which boulders are common occur quite frequently beside the road. Boulders which were rejected when gravel was taken out are also to be found in many old pits, such as those at Hawk creek, Middle camp, south of Mile 8 creek, and in the Kootenay valley.

The more convenient talus or morainic deposits, where stone for the crusher can be obtained, are the following two.

Near the road chainage 330+00, between Mile 8 creek and the big gravel slide at chainage 250 + 00, a talus on west side of the road. There, a certain amount of angular blocks of fresh limestone can be had without any excavating.

Opposite the road chainage 666 + 00 and approximately 75 feet west of the road, a big slide made up of apparently morainic material composed of limestone fragments and clay. A more detailed description of this deposit is given elsewhere in this report.

GENERAL CHARACTER OF THE GRAVEL

Good road gravel should not disintegrate under traffic, and the pebbles should be of variable size so as to have the minimum quantity of voids. A certain amount of fine material is necessary to fill these voids. Gravel to be used for road surfacing should not contain pebbles over 1½ inch in size and should have at least 55 and not more than 75 per cent of its constituents retained on a ¼ inch screen. From 8 to 15 per cent of the material under ¼ inch in gravel should be clay, oxide of iron or similar binding material. The binding or cementing value of gravel is also improved by an appreciable amount of calcium carbonate. It is claimed that if a gravel occurs in a somewhat cemented condition in the bank it is likely to make a

good road material, and that gravels containing many pebbles of rocks which have good roadmaking qualities are desirable. Specifications usually call for gravel containing not more than 15 per cent of soft, friable pebbles. Angular pebbles will compact under traffic more rapidly than rounded pebbles and produce a firm road surface in a shorter time. The fact that a gravel packs quickly does not necessarily indicate that it will make a good road, for clayey gravels do this, and those containing more than 20 per cent clay are liable to make a muddy road.

The gravels found in the Vermilion valley between Hawk creek and the Kootenay river differ somewhat in their general character from those occurring along the other sections of the road, which were surveyed in 1920 and 1921[1].

From Castle to the Summit the gravels to be found are generally coarse and carry a high percentage of sand with a small binder content. Those occurring between Marble canyon and Hawk creek vary from very sandy to loamy or clayey gravels. In the Sinclair pass the amount of fine material passing the 200 mesh sieve carried by the gravels is quite high and of a more silty nature.

In the area surveyed last summer the gravels of the various deposits lying along the Vermilion river resemble each other more in character and are as a rule of fairly good road-making quality. The gravels in the flats generally are loose and carry in the upper layer a certain amount of loam. Below a certain depth they lack binding material. In the ridges and in the exposures where the road cuts through shoulders of deposits the gravels, though not cemented, contain a sufficient amount of material in powder form to bind well in a road surface.

Gravel is especially plentiful near the Kootenay river and is of very good quality for road surfacing. In most deposits there, the gravel is fine and possesses good binding qualities. The proportion of sand is comparatively low, the percentage of voids in the total aggregate high, but the pebbles are coated with calcium carbonate, and because of the highly calcareous nature of the binder contained good results can be expected from their use. The gravels found in the bottom of the valley south of Kootenay Crossing, however, lack binding properties.

Thirty-five samples were collected from the more promising deposits between Hawk creek and McLeod Meadows camp. In the majority of these samples the proportion of sand to gravel is just about right. In twenty-six samples the percentage of sand is not over 40, and in four samples only is it less than 25. The amount of fine material passing the 200 mesh sieve contained in the sands ranges from 5 to 33 per cent, in eighteen samples this percentage being 10 or over. The amount of clay and silt in the sands, determined by washing through a 200 mesh sieve, was found to vary from 5 to 47 per cent, with fourteen samples having 20 per cent or over and eleven samples with 10 per cent or under.

Few of the gravel samples collected are clean enough to make good cement concrete, but washed gravel for that purpose can be easily found along the river.

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[1] Mines Branch, Summary Report for 1919. Report of the Road Materials Division, by K. A. Clark, p. 146. Report on Investigation of Road Materials in Rocky Mountains Park in 1921, by H. Gauthier, Mines Branch.

The petrological composition of the gravels tested does not vary greatly, as an examination of Table II will show. This composition includes only a few types of rock which were divided into durable, intermediate and soft types according to their average resistance to wear and impact. The durable types include igneous and metamorphic rocks, the intermediate types include limestone and dolomite, and the soft types include shales and weathered stones of all kinds. In nearly all the samples that were collected limestone pebbles predominate in number. In eleven out of thirty-five samples 75 per cent or more of the pebbles are of that type of rock. In six samples only the percentage of limestone pebbles is less than 50 per cent. Most of the gravels occurring north of Vermilion Crossing carry a high percentage of soft shales and weathered rocks, indicating that they will not wear well. The greater number of gravels examined south of Vermilion Crossing were, however, comparatively low in soft material, and so were the deposits along the Kootenay river. Of the total number of samples taken fourteen contain more than 20 per cent of soft pebbles, but in only four is this content higher than 50 per cent. In the majority of the gravels the pebbles are well rounded, but whenever shales occur in their constituents these are flat and thin in shape.

For purposes of comparison one sample of the road surface was collected at chainage 740, a short distance south of Vermilion Crossing, where the road subgrade, made up of gravel, had packed to a hard and smooth surface. This is sample No. 15 in the tables. It may be noted that this gravel is comparatively fine, well graded, and contains the amount of binder which is usually recommended for road gravel. It is also largely composed of durable material.

Among the gravel samples tested there are several which compare favourably with this material and should give satisfactory service on road surfaces.

In the following pages the character of gravels in the different deposits is briefly described, together with such information regarding their actual service as could be obtained from the road subgrade wherever they entered into its construction.

This section, and the tables giving the results of tests, should be read, for comparison purposes, in conjunction with the results of analysis of sample No. 15.

LOCATION AND DESCRIPTION OF GRAVEL DEPOSITS

Road chainage 447 + 00[1]

Immediately south of Hawk creek, ridge 50 feet wide extending eastward at right angle with the road.

Exposure in road cut, from 5 to 7 feet high and approximately 30 feet long, of loose, coarse, loamy gravel largely composed of rounded fresh limestone pebbles. One-half of the gravel is over 1½ inch in size but there are few boulders present. The sand, which is in the proportion of 33 per cent, is of medium grading and contains a large percentage of material passing 100 mesh sieve.

A high working face is not available but small quantities of fairly suitable gravel can be had for road maintenance work.

[1] Chainage from north.

Sample No. 1 was taken. The results of tests are given in tables I and II.

485 + 00 to 505 + 00.

From chainage 485 to chainage 505, some material could be had, but at a level lower than that of the road. The latter here is located on top of a flat topped gravel bank. The material exposed on the road grade and in the ditches consists of gravelly loam. In depth, loose coarse gravel with only little binder is to be found. This gravel contains a large proportion of shales and soft pebbles.

A 10 to 15 foot working face is readily available at various points along the edge of the bank, particularly opposite chainages 492 and 499, at distances of from 10 to 25 feet west of the road side. However, the use of this material is not to be recommended because it lacks binder and contains a high proportion of shales. Sample No. 2 was collected in the neighbourhood of chainage 488 + 00.

547 + 00

Lens-shaped deposit of small extent. Fairly well graded, medium gravel with clay binder, is exposed in a 5 foot cut, on east side of road, at curve. This is good binding gravel but the amount available is small. See sample No. 3.

555 + 00

On either side of spring creek, exposures of uneven, loose bouldery to sandy gravel. The chances for excavating are poor. Sample No. 4 was taken and tested.

572 + 00

Just south of outcrops of shales, at curve, there is a good exposure of fine gravel with binder. Few boulders present. There is possibility for a 10 foot working face but the amount available is probably not over 1,000 cubic yards. North of the rock outcrops fine blue gravel is to be seen but it is devoid of binder.

One sample of the material occurring south of the rock outcrop was collected and has tested very good, as far as grading and amount of binder present are concerned, but, on account of the very high proportion of soft flat shales contained, this gravel will not resist wear. See the results of analysis of sample No. 5 in the tables.

583 + 00

At top of grade, where road reaches a flat north of Middle camp. Cut 7 feet high and approximately 100 feet long on east side of road. The exposure consists of 1 to 2 feet of loamy sand underlain by gravel varying from fine to bouldery and containing a moderate amount of clayey binder.

There is a fairly good opportunity here for a pit. The deposit is in the form of a small ridge, the maximum height of which is about 15 feet above the road level. Probably a thousand cubic yards could be excavated.

Sample No. 6, representative of the average bank run, was taken and proved to be properly graded, to contain a sufficient amount of binder, and to be largely composed of fresh limestone pebbles.

The results of tests are given in the tables.

850 + 00 to 836 + 00[1]

Ridge parallel to road, on east side, averaging 20 feet in height, with rather steep slope only 25 feet or so away from the road side.

Three test pits were made. In a 5 feet deep excavation made in the upper part of the slope, opposite chainage 838, the material encountered was 2 feet of gravelly loam, with boulders, underlain by loose, well graded, fine gravel, carrying a small percentage of clay and silt. See sample No. 9.

The two other test pits, from 4 to 5 feet deep, were made at different levels opposite chainage 845.

The upper one, near the top of the ridge, revealed the presence of one foot of coarse loamy gravel overlying medium gravel and coarse sand with little binder. See sample No. 7.

In the lower pit, at the foot of the slope, coarser gravel with a high proportion of loam was found. See sample No. 8.

The results of laboratory tests upon samples Nos. 7 and 9 show that the binder content is low, but that the gravel is well graded and suitable for surfacing on clayey subsoil.

Large quantities of gravel could be easily had by excavating along the edge of this ridge, as a wall from 12 to 15 feet high and several hundred feet long could be obtained. The timber is light and the amount of necessary stripping not over 2 feet thick.

820 + 00 to 810 + 00

Flat topped ridge, presenting an escarpment about 15 feet high and from 20 to 25 feet distant from the road, on east side.

One test hole, 5½ feet deep, made in the upper part of the slope at chainage 812, showed that the material in the upper 2 feet is bouldery gravel carrying much loam, while underneath rather coarse gravel with coarse sand, containing very little binder, is to be found. One sample of this material was taken. See the results of analysis of sample No. 12 in the tables.

A similar test pit, 7 feet deep, was made opposite chainage 815. There, the material encountered is as follows: 2 feet of loamy coarse gravel underlain by 2 feet of loose very coarse gravel followed in depth by well packed gravel such as represented by sample No. 11. This material is well graded, free of boulders, and carries a certain amount of clay.

In a third test pit made at chainage 816, coarse gravel, with a large proportion of boulders between 3 and 6 inches in size, was found in the upper 3 feet. Below this, medium size gravel with only a small percentage of sand occurs. This gravel, however, contains a good deal of binder in the form of clay lumps and calcium carbonate coating on the pebbles, and therefore should bind well. Sample No. 10.

A 10 to 15 foot wall could be developed along the edge of the ridge between the above mentioned chainages, close to the road, and with but a comparatively small amount of stripping to do. The chances are that fairly good gravel will be found.

[1] Chainage from south, Kootenay Crossing.

755 + 00

From chainage 770 southward, there are several exposures of gravel subsoil but nowhere do the deposits form any elevation which would favour the opening of a pit. Some material, however, could be obtained by scraping off the surface in flats on either side of the roadway.

As a rule the gravel along here is fine but lacks binding material. An average representative sample was collected from the 2 feet deep road cut at chainage 755. Sample No. 13.

747 + 00

Vermilion Crossing. On either side of the crossing there are borrow pits from which material was obtained for the construction of the bridge approaches. The material exposed in the shallow excavation, alongside the road, at the north end of the bridge, consists of loose, clean, fine sandy gravel.

South of the river the exposure is 6 to 7 feet high, but the gravel is loose in the bank, coarse to bouldery, and lacks binder. Sample 14 was taken and tested.

740 + 00 to 705 + 00

From Vermilion Crossing southward to chainage 705, the natural subsoil is loamy fine gravel. Certain stretches of the newly graded road were found to be in good condition, while in places, where loam is in excess, mud holes and deep ruts had formed.

For purposes of comparison, one sample of the road surface was collected at chainage 740, where the road grade had packed to a hard and smooth surface. See the results of analysis of sample No. 15 in the tables. It may be noted that the material is comparatively fine, well graded, and contains the amount of binder which is usually recommended in the specifications for road gravel. Among the gravel samples collected there are several which compare favourably with this material. Therefore, the gravel which they represent ought to give as good results if used in road surfacing.

Except very low ridges located a short distance west of the road, between chainages 740 and 735, and in the neighbourhood of chainages 725 and 711, the flat character of the country offers no opportunity for a pit. The underlying gravel is exposed in the high bank of the river to the east of the road, but, as a rule, it is sandy and does not contain binder. A pit there would not be practicable as such location would mean a difficult haul up hill to the road.

A representative sample of the well graded fine gravel occurring at the surface along this section was taken at chainage 705 where the road cuts through the edge of the small knoll. The exposure shows clay, which if mixed in proper proportion with the overlying gravel would improve it. The amount of gravel available there, however, is small.

Sample No. 16.

666 + 00

Big slide approximately 75 feet west of road, composed of apparently morainic material consisting of fresh limestone angular fragments and clay.

In a 4 feet deep test pit dug in the steep and cleared slope, facing the road, the material encountered was 2 feet of coarse material underlain by finer aggregate with a high clay content but which appears not to be in excess.

The analysis of sample No. 17, which was taken at a depth of 4 feet, at a point about 20 feet above the flat, shows that this aggregate is well graded and suitable for road surfacing. Its use, as a trial, is advisable.

If the deposit runs uniform, very large quantities of material can be obtained without trouble, as a high working face and only a small amount of stripping are possible features.

In case only small quantities of material are wanted, a still better aggregate will be found in the occurrence described below.

The results of tests upon sample No. 17 are given in the tables.

660 + 00 to 653 + 00

Morainic material, similar to that described in the preceding number but of a still better grading, is exposed in extensive cuts made to reduce the grade of road and to build a fill.

The sections of exposure vary from a few feet to 15 feet in height, and large quantities of material could be easily excavated, with hardly any stripping to do.

Sample No. 18, which was collected opposite chainage 653, shows that in this mixture of limestone fragments and clay the stone aggregate is well graded, and that the clay content is about just sufficient to act as binder.

The actual condition of the road surface where this material occurs, after it had been built for a month, last summer, proved that the clay content is not too high.

An inspection, in the spring, would probably give better information as to the suitability of this material for surfacing.

See sample No. 18, in the tables.

651 + 00

Immediately south of the occurrence above described, gravel is exposed on the west side of the road in a cut 8 feet high and some 30 feet long. This gravel varies in texture from fine to medium and contains comparatively little sand. The sand is coarse and carries some binder. Some of this gravel was used last summer for maintenance. The amount available is small.

Sample No. 19.

648 + 00 and 630 + 00

Interstratified fine gravel and clay is exposed in cuts varying from 5 to 10 feet in height at chainages 648 and 630. In the first instance the gravel is coarser and contains more sand than at chainage 651. In the second deposit good fine gravel occurs but with large pockets of clay. The run of the bank could not be used, as an excess of clay present was shown by the muddy surface of a section of the road grade built with it.

The amount available is small.

Sample No. 20.

621 + 00

Road cut 100 feet long and from 4 to 8 feet high, showing 2 feet of sandy loam underlain by coarse gravel without binder. An 8 foot wall can be excavated and large quantities of gravel are available. Unless used on clayey subsoil this material will not yield a bond.

Sample No. 21.

608 + 00

Coarse gravel, resembling in character that exposed at chainage 621, is to be found in road cuts, 4 to 6 feet deep, between chainages 608 and 606. See sample No. 22.

593 + 00 to 576 + 00

Important ridge, in brule, rising to over 35 feet above the road and close to it, on the west side.

Test holes have shown that this deposit is very complex in its composition. Fine sandy gravel underlying gravelly loam occurs in place, while elsewhere stony clay is to be found.

The chances are that material similar to samples No. 24 and No. 25, composed entirely of limestone fragments and clay, properly graded, may be found in this ridge.

One sample of the surface gravel occurring along the roadside in the vicinity of chainage 580 was taken. See sample No. 23. It is medium size gravel containing only a low per cent of sand and very little binder.

Sample No. 24 was collected from the road cut at chainage 576, and although containing a high percentage of stone aggregate over 2 inches in size this material is well graded and contains just enough clay to make it yield a good hard road surface.

570 + 00 to 568 + 00

Drift material like that occurring at chainage 576 is exposed in cuts on west side of the road.

The aggregate, here, contains more clay than that of the above mentioned, but quite large quantities of fairly suitable material are easily available.

Sample No. 25, which was collected and tested, represents about the average character of the deposit.

547 + 00 to 544 + 00

Exposure in road cut, of medium size gravel with a low per cent of sand and very little binder.

A 6 foot wall alongside the road could be excavated.

The use of this gravel, however, is not recommendable as it lacks fines and binding material.

See sample No. 26.

510 + 00 to 500 + 00

Loose gravel varying from fine to medium in texture, in which the proportion of sand is comparatively low, but with a fairly high content of clayey loam binder is exposed in a series of cuts on the road. This is near where the road reaches the flat of the valley.

The loamy gravel occurs at the surface of slopes and the thickness is variable but in no case great. It is underlain by sandy gravel without binder. However, there are good chances for fairly large quantities of gravel with binder. Sample No. 27 was collected in the neighbourhood of chainage 501.

450 + 00 to 403 + 00

Loamy, fine to coarse gravel subsoil is frequently encountered in the flat, from chainage 450 + 00 to Mile 8 creek, a distance of nearly one mile.

Some gravel was obtained from a shallow borrow pit at chainage 430. More gravel with loamy binder could be had by scraping off the top soil at various points along this stretch of the road. The timber, as a rule, is light, and the overburden to be stripped amounts to very little.

Sample No. 28 is representative of the average character of the gravel to be found here. The road surface showed that this material packs well.

Sample No. 28.

400 + 00

A few hundred feet south of Mile 8 creek, at first curve, there is a good exposure of drift material which presents good opportunity for opening a pit.

The exposure consists of a 15 foot cut. The material is composed of angular, flat, mostly limestone pebbles and contains clayey binder. In the upper 3 feet the aggregate is quite coarse, but the balance of the cut shows fine to medium textured material. This is fairly suitable road material and it could be easily excavated. The amount available is in the neighbourhood of a thousand cubic yards.

Sample No. 29 was collected and tested.

400 + 00 to 250 + 00

From the above described occurrence to the big slide at chainage 250, the subsoil generally encountered, in the flat, is either fine loamy gravel or sandy washed gravel. There is no ridge close to the road from which material could be had. However, there are several borrow pits where some of this surface gravel has been excavated to a shallow depth for grading purposes.

The gravel to be found in these pits is represented by sample No. 30, which was taken from a pit at chainage 305. It lacks binding material.

250 + 00

Big, steep slide over 75 feet in height, with extensive exposure of stratified gravel, sand, clay and silt.

The gravel, as a rule, is fine in texture and silty, but certain parts of the section exposed at a point where the river has cut the deposit show well cemented gravel. The amount of boulders present in the deposit is comparatively small.

The quantity of good gravel available is very large, the only drawback to this deposit being a too high and dangerous face. The road lies at the foot of the slide and the gravel could be loaded directly into the trucks by the use of chutes.

The actual condition of the road alongside this slide showed that this gravel packs to a hard and smooth surface. One sample was taken from the lower part of the wall at its northern end. It represents the average character of the greater part of the material exposed there.

See the results of analysis of sample No. 31.

Similar gravel is also to be found in a big cut, on west side of the road, between chainages 255 and 260. The conditions there may be found more convenient for excavating.

195 + 00 to 190 + 00

Fine gravel containing binder is exposed in hillside cuts on the west side of the road. This occurrence offers a good opportunity to obtain gravel of good quality. The pebbles in this gravel are well graded in size and are coated with clay and calcium carbonate which cement them together. The boulders present in the exposure are few. Large quantities of good gravel can be obtained directly from the road side with only a small amount of stripping. A 20 foot working face can easily be made.

The road surface along this hillside cut is well compacted, firm and smooth.

Sample No. 32 was collected and examined. See the tables.

154 + 00

Gravel pit, from which material was obtained for the surfacing of that section of the road immediately north of this occurrence where sand is the natural soil.

The pit is located on the edge of a knoll on the west side of the road. A 15 foot wall has been developed. The gravel is fine textured, fairly uniform, and contains very little sand. Pockets of fine sand, however, occur in the exposure. The gravel is fairly well cemented in the bank, the pebbles being coated with clay and calcium carbonate. However, probably because of its low sand content, this gravel has failed to compact on the road. The road surface, where it was used, is loose and ravels. If used on clayey subsoil, this gravel will give much better results.

The amount of gravel still available in this deposit is large.

Sample No. 33 was taken and tested.

130 + 00 to 90 + 00

Between chainages 130 and 120, from 2 to 3 feet of loamy fine gravel overlying loose, clean, fine gravel without binder, is exposed in several instances, in cuts varying from 3 to 4 feet high, along the road.

This surface gravel packs well on the road but the topography in the immediate neighbourhood, which is flat, offers no facility for excavating except by scraping off the surface.

Similar material is also exposed in cuts from 4 to 8 feet high at chainages 105, 95, and 90.

85 + 00 to 0 + 00

From chainage 85 or so, southward to Kootenay crossing, a distance of approximately one mile and a half, occurrences of very good road gravel with binder are plentiful. The gravel runs from fine to medium in texture, is packed hard in the bank, and well cemented with clay and calcium carbonate coating the pebbles.

Among the best occurrences are the following:—

Around the big bend in the neighbourhood of chainage 85. There, 8 feet of the gravel is exposed in an extensive cut. While widening the roadway at this dangerous curve, very good material will be obtained by excavating.

Between the chainages 75 and 70—

From 5 to 8 feet thick of fine gravel carrying enough binder is readily available from the road side.

From chainage 65 southward to small lake, chainage 50—

All the way very suitable fine gravel, containing binding material, is almost continuously exposed in cuts varying in height from a few feet to 10 feet. If a site for a huge pit is looked for, a knoll rising to over 50 feet in height, with steep slope and almost cleared of timber, just north of the lake, will be found. A large pit could be operated on the flank of this deposit in the vicinity of chainage 52, that is about one mile north of Kootenay river.

Other occurrences which offer good opportunities for the opening of pits are the extensive road cuts which face the two small lakes at chainage 45 and 30. The gravel in these deposits does not carry as much clay and loam as that occurring north of the lakes, but it is well cemented. The pebbles are coated with a limy substance which produces a good bond. In either place a large amount of gravel can be obtained without trouble.

One sample of this gravel was collected from the road cut along the first lake north of the Kootenay river. It is sample No. 34 in the tables.

The suitability for road surfacing of the gravel to be found along this section of the highway is well shown by the actual condition of the road surface, made of the same. This surface, as a rule, is well bound.

Samples were also taken for purposes of comparison, from the pits which were opened last year at chainage 10 + 00, a short distance north of the Kootenay, and at the end of the tangent south of the river. They are samples No. 35 and No. 36. In examining the results of analyses in the tables, it will be noted that they contain a small percentage of sand. These gravels belong to deposits which have once been washed. However, they carry a sufficient amount of binder to give good results in a road surface, their constituents being thickly coated with carbonate of calcium, which will act as a cement.

The gravel in both pits is of rather uniform and fine grading and fairly well packed in the banks. The upper foot carries loamy clay.

Not less than one thousand cubic yards of material have been excavated in the pit north of the Kootenay, but several times as much material is still available. The deposit forms a big knoll.

In the other pit located at the sharp turn of the road, to the south, in the Kootenay valley, more material can be obtained, but there the gravel occurs in a flat and a high wall could not be excavated.

0 + 00 to 243 + 00[1]

Between Kootenay crossing and Mud creek, in the Kootenay valley, gravelly soil is of frequent occurrence. However, on account of the flat character of the country, there are few opportunities for the opening of

[1] Chainage from north, 0+00 at Kootenay Crossing.

gravel pits. The more suitable sites are the extensive cut, on hillside, about half way between the river crossing and Mud creek, and a ridge located at about 150 feet west of the road and running parallel to it, a short distance north of the creek.

In the first instance there is a good exposure of fine clayey gravel. A high wall is readily available to start with in excavating, and large quantities of gravel can be obtained without trouble. The material exposed resembles that found to the north of the Kootenay near the lakes. It is packed hard in the bank and the pebbles cemented with clay and calcium carbonate.

In the second instance, the chances are that good clayey gravel will be found. Test holes made along the foot of the slope showed the presence of fine gravel with clayey binder. There is a clearing from the road to the foot of the ridge and the latter is but slightly covered with shrub and light timber.

250 + 00

From Brothers camp to McLeod Meadows, where our survey ended in 1921, there are several gravel pits from which material was obtained for the surfacing of the road last year.

These gravels, as a rule, resemble each other. They are generally fine in texture, fairly well graded, but except in the upper part of the deposits, do not carry enough binder. The road surface built during last summer appeared to be in excellent condition last fall, but, although this surface was rolled, it is doubtful whether under heavy traffic it will stand up without ravelling.

For purposes of comparison, one sample was collected from the pit, located immediately west of the road, north of Brothers' camp. It represents the average character of the gravel occurring in this part of the valley. In examining the results of analysis of sample No. 37, it will be noted that the per cent of clay and silt is rather low, but this may be counterbalanced by the presence of calcium carbonate on the pebbles.

650 + 00

Approximately 2 miles south of McLeod Meadows camp, there is an important ridge of drift material, parallel and close to the road. The ridge is several hundred feet in length, over 25 feet in height, and presents an escarpment facing the road, a short distance south of a creek.

A small pit was opened there, at a late date last year, when the surfacing of that section of the road had been nearly completed. A small amount of the material excavated was tried as surfacing material and the best results were obtained. This drift material is composed of gravel compacted very hard in the bank and cemented with highly calcareous clay. As a road surfacing material, the run of the bank is too coarse, about 30 per cent of the gravel being coarser than 1½ inch. The material under this size, however, is well graded and contains a proper amount of binder. It packs well on the road and gives a smooth and hard surface.

The amount available is large and a high wall can be excavated with little stripping to be done.

Sample No. 38 was collected from the actual wall of the excavation just started. The results of the tests are given in the accompanying tables.

TABLE I

Results of Physical Tests on Gravel Samples from the Hawk Creek-McLeod Meadows section of the Banff-Windermere Highway, Rocky Mountains Park

Sample No.	Location of deposit (road chainage)	Granulometric analysis														Per cent passing 200 mesh	Remarks on suitability for use in gravel road construction
		Proportion of gravel to sand		Gravel							Sand						
				Per cent retained on screens							Per cent retained on sieves						
		Per cent grav.	Per cent sand	2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200	
1.....	447–00[1]	67	33	20	5	25	15	6	11	18	24	15	10	9	12	16	14	Too high proportion of gravel over 1½ inch in size.
2.....	488–00	50	50	17	5	12	19	15	12	20	25	22	19	15	8	3	8	This material should prove satisfactory in the base course of gravel roads. Lacks binder and is too coarse for top course.
3.....	547–00	55	45	8	10	17	20	11	18	16	20	11	10	10	9	20	20	This material should prove satisfactory.
4.....	555–00	60	40	8	3	11	20	16	17	25	20	13	11	10	8	25	13	
5.....	572–00	55	45	....	4	13	18	17	19	29	22	16	13	12	10	14	13	Well graded fine gravel and sand.
6.....	583–00	66	34	....	5	11	15	16	23	30	38	17	11	9	6	9	10	This material should prove satisfactory.
7.....	845–00[2]	60	40	....	15	9	16	14	18	28	23	25	25	12	5	3	7	Per cent of fines is low.
8.....	845–00	60	40	....	29	13	12	14	14	18	21	13	10	8	6	9	33	High proportion of pebbles over 2 inches.
9.....	839–00	50	50	....	....	14	18	14	23	31	23	9	9	22	16	14	7	Well graded fine gravel. Should prove satisfactory on clayey subsoil.
10.....	816–00	80	20	18	21	22	28	5	3	3	17	20	18	10	9	20	6	This gravel is coarse and the proportion of sand is low. Lacks material between 1 inch and ¼ inch in size.
11.....	815–00	70	30	....	8	12	29	13	15	23	34	19	15	8	4	5	15	Should prove satisfactory.
12.....	812–00	57	43	20	6	13	18	8	11	24	35	27	18	6	2	4	8	Large proportion of gravel over 2 inches. Low per cent under 28 mesh.
13.....	755–00	58	42	....	27	12	22	10	14	15	15	16	27	28	6	2	6	Should prove satisfactory on clayey subsoil only.
14.....	747–00	65	35	14	11	12	20	10	13	20	29	21	19	15	7	3	6	" " " "
15.....	740–00	57	43	....	6	17	17	9	16	35	18	11	15	20	9	9	18	Sample from firm and smooth road surface for comparison purposes. Notice grading and amount of fine.
16.....	705–00	63	37	....	9	14	22	17	16	22	30	23	19	13	6	3	6	Well graded gravel but lacks binder.
17.....	666–00	45	55	....	14	15	22	10	15	24	16	13	13	10	8	18	22	Proportion of sand a little high, but on account of high clay content should bind well.
18.....	653–00	65	35	....	16	17	25	9	13	20	25	15	12	11	9	8	20	This material should prove very satisfactory.
19.....	651–00	63	37	....	17	4	18	16	18	27	31	16	14	13	8	5	13	Well graded.
20.....	630–00	55	45	....	17	10	12	13	18	30	28	15	13	13	9	9	13	"

21.....	621–00	64	36	27	4	11	23	12	9	14	16	20	36	19	4	2	3	Coarse gravel without binding material. Suitable for concrete.
22.....	608–00	70	30	5	11	13	23	13	15	20	24	24	22	15	5	4	6	Coarse gravel without binding material. Suitable for concrete.
23.....	580–00	75	25	9	7	12	29	11	16	16	23	9	9	23	18	7	11	The proportion of sand is low.
24.....	576–00	65	35	32	16	7	13	8	13	11	17	15	13	12	9	16	18	This material should prove satisfactory, after the excess of material over 2 inches has been rejected.
25.....	568–00	60	40	18	16	12	10	7	17	20	19	15	13	12	9	16	16	This material should prove satisfactory, after the excess of material over 2 inches has been rejected.
26.....	545–00	77	23	8	13	20	25	12	11	11	20	15	19	24	10	4	8	The proportion of sand is low.
27.....	501–00	75	25	15	7	20	20	12	13	13	22	11	8	15	18	10	16	Although containing a low per cent of sand may prove satisfactory on account of binder contained.
28.....	430–00	75	25	7	11	29	19	11	11	12	18	14	21	19	7	5	16	Although containing a low per cent of sand may prove satisfactory on account of binder contained.
29.....	400–00	60	40	16	4	12	20	12	15	21	30	18	12	8	5	7	20	About 20 per cent of the gravel portion to be rejected as being too coarse, but balance well graded.
30.....	305–00	67	33	....	12	6	28	15	17	22	17	13	20	34	10	3	3	Suitable for concrete.
31.....	250–00	75	25	....	8	24	29	15	13	11	13	9	9	13	20	26	10	Sand silty but on account of low proportion of sand to gravel will prove satisfactory.
32.....	190–00	62	38	....	7	8	25	12	16	32	24	12	10	10	8	28	8	Well graded. Should prove very satisfactory.
33.....	154–00	84	16	....	5	4	11	13	18	49	42	8	5	6	7	22	10	Well graded but too low proportion of sand. Reason why this material, though containing binder, failed to pack in roadbed.
34.....	30–00	55	45	12	12	5	14	8	16	33	33	24	16	8	3	8	8	This material has proved very satisfactory.
35.....	10–00	75	25	....	....	18	27	17	18	20	26	12	17	20	8	10	7	Fine gravel with too low per cent of sand. Has proved satisfactory on account of binder contained.
36.....	0–00	80	20	6	17	16	9	13	20	21	58	17	6	3	2	4	10	Fine gravel with too low per cent of sand. Has proved satisfactory on account of binder contained.
37.....	250–00[3]	65	35	4	8	8	15	10	15	40	36	20	20	10	3	2	9	Well graded, but lacks binder.
38.....	650–00	60	40	15	4	11	15	13	17	25	28	15	14	20	10	6	7	Except an excess of material over 2 inches is well graded. Service test has given very good results.

[1] Chainage from north.
[2] " south, Kootenay Crossing.
[3] " north, "

TABLE II

Results of Physical Tests on Gravel Samples from the Hawk Creek-McLeod Meadows section of the Banff-Windermere Highway, Rocky Mountains Park

Sample No.	Character of material						Remarks
	Composition of pebbles[1]			Shape of pebbles	Impurities	Per cent[2] clay and silt in sand	
	Durable	Intermediate	Soft				
	Per cent	Per cent	Per cent				
1	5	90	5	Rounded	$CaCO_3$ coated on the pebbles and organic matter	28	This material should bind well
2	5	40	55	Rounded and flat		10	Too high content of soft shales. Lack of binder
3	.....	75	25	Angular, flat	Organic matter and clay	39	This material should bind well
4	20	20	60	Angular	" "	34	High proportion of soft rock. Should bind well but will wear fast
5	15	15	70	Angular, flat	" "	29	High proportion of soft rock. Should bind well but will wear fast
6	.....	95	5	Flat	Clay	18	Composed of fresh limestone pebbles with sufficient amount of binder
7	10	50	40	Rounded and flat	Little $CaCO_3$	10	Low binder content
8	15	30	55	" "	Some $CaCO_3$ organic matter and loam	37	High proportion of soft rock. Loamy gravel, should bind well
9	20	50	30	Subangular	Little organic matter	14	
10	45	50	5	Rounded	$CaCO_3$ and clay coating in large amount	47	High per cent of clay in sand, but it will be noted that the gravel carries only 20 per cent of sand
11	45	45	10	Rounded to subangular	$CaCO_3$ in small amount on the larger pebbles	18	Should bind well
12	40	60	.....	Rounded		10	Low binder content
13	20	80	.....	Rounded to subangular		10	" "
14	5	95	.....	" "		10	" "
15	40	50	10	Rounded	Organic matter and loam	23	Sample from firm and smooth road surface
16	25	50	25	"	Very little $CaCO_3$	10	Low binder content
17	.....	100	.....	Angular	Clay	45	High clay content in the form of lumps
18	.....	100	.....	"	"	28	Proportion of binder to total aggregate about proper
19	15	65	20	Subangular to rounded	Clay and organic matter	13	
20	.....	95	5	Rounded and flat	" "	33	Should bind well
21	20	50	30	Rounded		5	Lacks binder

22	25	75	.....	Rounded to sub-angular	.....	5 Lacks binder
23	10	60	30	" " ..	Loam.....	12
24	.....	100	.....	Angular.....	Clay.....	25 Should bind well
25	.....	100	.....	Subangular.....	"	40 High clay content, may produce a muddy surface when wet
26	20	65	15	Rounded.....	$CaCO_3$ in small quantity on pebbles	10 Low binder content
27	55	25	20	Rounded to sub-angular	Loam and organic matter.....	20 Should bind well. Loam binder
28	25	65	10	" " ..	" "	15 " " "
29	.....	75	25	Angular and flat....	Clay.....	25 " " "
30	20	50	30	Flat.....	.....	5 Lacks binding material
31	.....	95	5	Rounded.....	.....	20 Binder of rather silty than clayey nature
32	40	50	10	"	$CaCO_3$ organic matter and clay.	35 Should bind well
33	35	55	10	Rounded to sub-angular	$CaCO_3$ and clay coated on pebbles	24 Proper amount of binder but sand content too low
34	25	70	5	" " ..	$CaCO_3$ coated on pebbles and highly calcareous clay	16 Should bind very well. The amount of clay present is not high, but calcium carbonate coating present to act as binder
35	40	50	10	" "	$CaCO_3$ coated on pebbles and highly calcareous clay	15 Should bind very well. The amount of clay present is not high, but calcium carbonate coating present to act as binder
36	10	50	40	Rounded.....	$CaCO_3$ coated on pebbles and highly calcareous clay	15 Should bind very well. The amount of clay present is not high, but calcium carbonate coating present to act as binder
37	15	60	25	"	Thin film of $CaCO_3$ on pebbles..	7 Doubtful if will bind without rolling
38	20	75	5	"	" " " ..	15 Service test has proved that this material binds well

[1] The pebbles are classed as durable, intermediate or soft, according as they are composed of rock which would show a per cent of wear of less than 3, between 3 and 6, or over 6, respectively. The durable class, here, includes pebbles composed of fresh rocks of igneous origin, the intermediate includes fresh limestone, and the soft includes weathered pebbles of both types and shales.

[2] This determination was made by washing through a 200 mesh sieve.

III

PROSPECTING FOR ROAD MATERIALS BETWEEN MASSIVE AND JOHNSON CANYON

During the first part of the field season a successful prospecting was carried on between Massive and Johnson creek along the Banff-Lake Louise road, where no suitable material for road work had previously been found.

The general character of that area, called Hillsdale, is very rolling. The road for 1½ mile cuts through a series of small foothills made up of decomposed shales. On either side of this hilly area, near Johnson creek and near Massive, gravel occurs, but it lacks binding material.

A thorough examination of every hillock within hauling distance of the road and the digging of many test holes finally resulted in the finding of gravel at two places close to the road and in sufficient amount to constitute a supply of material for the surfacing of that stretch of the road.

At a distance of 1·4 mile east of Johnson creek, gravel was found on the crest of a ridge immediately north of a shallow gully on the north side of the road. The maximum thickness of this cap-like covering of gravel may exceed 20 feet. The ridge presents steep slopes and is only partly covered with light bush. It is continuous for 500 feet in a direction parallel to the road and its elevation above the gully is about 100 feet.

Two test pits, four feet deep, were made. The gravel found in them is fine in texture and carries only a small proportion of stone over two inches in size. The proportion of material passing the ¼ inch screen is rather high but contains a good deal of binding material. The pebbles are coated with calcium carbonate, angular in shape, and largely composed of fairly durable shaly limestone. Soft or weathered pebbles are few. The material is packed very hard and ought to bind well on a road surface. The overburden consists of one foot of loose, gravelly loam.

The amount of gravel available can be estimated at several thousand cubic yards. The material could be loaded by means of a chute directly into trucks standing at the foot of the slope by the road side.

One sample was taken and analysed in the field. The proportions in the sand were determined roughly by volumetric means. They are as follows:—

Gravel, 33 per cent; sand, 57 per cent.

Sand retained on	8	mesh	30	per cent
" "	20	"	25	"
" "	48	"	15	"
" passing	48	"	30	"
			100	"

The second occurrence of gravel that was located lies at 1·7 mile east of Johnson creek, or one-half mile west of the large cribwork construction.

In following a gully, which is crossed by the road at the above mentioned mileage from Johnson creek, clayey gravel was found in small knolls at a distance of 100 yards north of the road. The amount available, however, was found to be small and the location unsuitable for a pit.

Similar material was finally located in a flat-topped ridge about 35 feet high, just east of this gully and immediately north of the road.

The chances are that fairly large quantities of gravel can be obtained there. It is difficult, however, to estimate the amount available without determining the average thickness of the gravel overlying the shale formation. But even if there should be a thickness of only a few feet, the opening of a pit will probably be worth while. A high wall can be developed next to the road, with very little necessary stripping. There is one foot of loamy overburden and a few trees.

The material taken out of a test pit 4 feet deep, made in the upper part of the slope facing the road, consisted of clayey gravel, packed hard and containing few boulders. The pebbles are largely composed of limestone and coated with calcium carbonate. On account of a reasonable amount of binding material contained therein this material should prove satisfactory in road surfacing. The analysis of one sample taken is as follows:—

Gravel, 36 per cent; sand, 64 per cent.

Sand retained on 8 mesh	35 per cent
" " 20 "	20 "
" " 48 "	20 "
" passing 48 "	25 "
	100 "

Deposit of Bedrock east of Johnson creek

About three-fourths of a mile east of Johnson creek, there is on the north side of the road an important outcrop of limestone, from which large quantities of material of good quality could be obtained for the production of crushed stone.

The deposit forms a high cliff several hundred feet long, parallel to the road and only one hundred feet or so distant from it. The total height of the scarp is approximately 75 feet, and about two-thirds of it is talus covered.

The beds exposed in the upper part could be easily quarried on a large scale.

The limestone is light grey, of medium texture, and occurs in thin beds which are frequently jointed and much fractured. The stone is somewhat weathered but laboratory tests performed upon one sample which was collected have proved that it compares favourably in durability with the Mount Rundle limestone, which is giving satisfactory results as road material in Banff. The physical properties of both limestones are given below:—

Sample	Per cent of wear	French co-efficient of wear	Tough-ness	Hard-ness	Specific gravity	Absorption in lbs. per cu. ft.
Limestone, east of Johnson creek	2·4	16·7	7	14·9	2·72	0·46
Limestone from Mount Rundle, Banff	3·3	12·3	6	14·9	2·72	0·50

IV

EXPERIMENTS FOR INVESTIGATING THE TEST FOR THE CRUSHING STRENGTH OF ROCK

Samples of granite for pavements were received in 1920 in connection with the compression test of rock which Dr. K. A. Clark, former chief engineer of the Road Materials Division of the Mines Branch, had in charge as Chairman of the Sub-Committee of Road Materials of the American Society for Testing Materials.

The sample consisted of two types of granite blocks respectively from Georgia and from Maine, the sample from the Pine Mountain quarry at Lithonia, Ga., being a light grey granite of fine texture, and that from the state of Maine being of a darker colour and of coarser but more even texture.

The object of the test was to determine the best type of test piece considered both from the standpoint of uniformity of results and of practicability in preparation. The experiments were carried on in cooperation with Mr. H. H. Scofield, Professor of Materials, College of Civil Engineering, Cornell University, Ithaca, N.Y.

The types of test piece considered were:—

Cylinder 1″ in diameter and 1″ high
" 1″ " " 1½″ "
" 1″ " " 2″ "
" 2″ " " 2″ "
" 2″ " " 3″ "
" 2″ " " 4″ "

Rough cube 2″ x 2″ x 2″.

The test pieces were prepared in accordance with the methods described in Bulletin No. 347 U.S. Department of Agriculture, 1916, by Frank H. Jackson "Methods for the determination of the physical properties of road building rock."

The cylinders were obtained from the block with a diamond core drill, cut to length with a diamond dust saw, and ground true on a grinding lap with the use of finely powdered carborundum as an abrasive agent.

The cubes were cut from the blocks with a chisel and ground to size in the same manner as the cylinders.

The test pieces were broken in an Olsen, 20,000 lbs. "3 Screw Type" Universal Testing Machine electrically operated. The speed used in applying the load was ·022″ per minute.

Every test piece was checked as to the dimension and the condition of the bearing surfaces before testing.

It was found that in the case of the 1″ cylinders it had been difficult in the preparation of the test piece to obtain accurately parallel faces.

The two types of granite used in making this test proved to be of similar quality. If stone blocks containing less cleavage planes and of a more uniform texture had been used, probably more comparable results would have been obtained in this test. More illuminating data would also have been secured if one of the samples used had been decidedly weaker than the other. The results of tests were sent to Mr. Scofield. They are not given here because they are part of a series of tests not yet concluded.

V

ROAD MATERIALS IN NOVA SCOTIA

R. H. Picher

Investigation on road materials in Nova Scotia, commenced at the end of the summer of 1920, was continued during the field season of 1922. The field work consisted of making an exhaustive study of the condition of the surfaced main roads, and inspecting and sampling materials available for road construction, especially along the lines of proposed improvements.

In order that the results of the investigation might be of most direct assistance to the Department of Highways of Nova Scotia, inspection work was carried on in those parts of the province where information on road materials was more urgently needed. After a consultation with Mr. W. A. Hendry, chief engineer of the Nova Scotia Highway Board, it was decided to begin first with the areas in the vicinity of the main highways. It is the policy of the present Department of Highways to improve all the main highways and maintain them in good condition at all times, and in this connection it was noticed that considerable progress had been accomplished in the last two years.

The following main roads were examined in detail, and all possible sources of surfacing material thoroughly investigated: The Truro-Sydney road; the South Shore road, from Halifax west to Shag Harbour; the Middleton-Bridgewater road; the Kentville-Chester road; the Windsor-Halifax road; the Windsor-Truro road, from Windsor to West Gore; the Annapolis Valley road, from Cambridge to Brickton; the Amherst-Parrsboro road. Some time was spent studying the character of extensive trap rock deposits outcropping for a distance of over one hundred miles along the bay of Fundy, and a few days were spent along the Amherst-Oxford road, to locate surfacing material. Sixty-five samples were collected and sent to the laboratory for testing.

ROAD CONDITIONS

The purpose in studying the condition of the surfaced roads was to ascertain the possibility of the materials employed in their construction to form a hard, resilient and impervious surface that can withstand the wear and impact of the traffic, and keep the water from penetrating into the subbase. By comparing results observed on roads with those obtained in laboratory tests, the suitability of materials for road purposes can be determined with reasonable accuracy. In making observations on the condition of a surfaced road its age and the methods used in its construction have to be taken into account. In cases where the material is spread over the road in one single course and left to be rolled by the traffic, no satisfactory conclusion can be drawn as to the wearing quality of the stone. A soft material will soon compact into a smooth surface under traffic, but will wear out very rapidly. A hard stone will remain loose a long time, so that two deep ruts will form in the wheel tracks before the stone is sufficiently compacted. Water will be retained in the depressions, thus softening the surface and accelerating its destruction. The amount of traffic is another important factor to be considered. A stone may have given good service for years on a light trafficked road and utterly fail under

increasing traffic. Some of the gravel stretches along the Halifax-Truro highway illustrate this. Another cause of failure in road pavements is insufficient drainage of the subsoil. A good gravel or broken stone surface will eventually yield under the weight of traffic, if the subsoil is not kept properly drained. Some clay soils in particular lose much of their supporting power when in a wet state.

Sydney-Truro highway.—The road between Sydney and Blackett Lake, gravelled in 1921 with material from pit No. 1, (see tables II, III), was in very good condition in the summer of 1922. The North Sydney road, surfaced with the same material and subject to a larger amount of traffic, showed slight signs of wear. A short stretch northeast of St. Peter surfaced with gravel No. 7 was in excellent condition. Much gravel from pit No. 11 has been used at different times to cover stretches of the main road between Heatherton and Antigonish. The road was in very good condition only on stretches not more than one year old. The Telford-Sutherland River road, surfaced with conglomerate gravel No. 16, and the Sutherland River-New Glasgow road, with gravel No. 14, were both in very good condition after one year. Gravel No. 16 is much too soft and No. 14 somewhat fine. The road at Alma has been kept in fair condition by gravelling at different times with material from deposit No. 17 and others. The gravel is not durable and the road requires frequent attention.

There are several sections surfaced recently, which have not been long enough in service to show any results as to the wearing quality of the material, and sections which have not yet been surfaced.

Truro-Halifax highway.—Due to the ever increasing amount of traffic much difficulty has been experienced in keeping this road in good condition. The Halifax-Stewiacke section was in course of repair during the summer. None of the materials found along this section are durable enough, taking into consideration present traffic conditions. Imported material will have to be used between Truro and Stewiacke, as has been done on the Shubenacadie-Enfield section, which was surfaced with gravel from deposit No. 20, with very good results. From Brookfield south, the roadbed, largely composed of clay, is very soft when in a wet state, and in order to maintain a good surface, thorough drainage is necessary.

The Enfield-Halifax section was in very good condition, except stretches between Waverley and Bedford, just completed at the end of the summer. On these most of the stone was loose. The road has been surfaced nearly all the way from Enfield to Halifax with crushed quartzite boulders. In places some fine material, largely fine sand and silt from the drift deposits, has been mixed with the crushed stone, so as to act as a binder and make a smoother finished surface. Stretches treated in this way wear much more rapidly than where crushed quartzite alone has been used. From Bedford to Halifax, the road has been surfaced with crushed quartzite and bitumen, and is one of the finest examples of bituminous macadam pavement.

Halifax-Shag Harbour highway.—This road carries a fairly large amount of automobile traffic as far as St. Margaret and light traffic farther west. The Halifax-St. Margaret section, surfaced with crushed granite boulders mixed with sand, was completed in 1920. It was in good con-

dition in 1922 but showed decided signs of wear. The St. Margaret-Chester section, covered in 1921 with gravel and sand largely composed of granite particles, was in rather poor condition and uneven after one year's service, due to the material being too fine. From Chester to Shag Harbour stretches have been surfaced at different times. The road is in good condition where the surfacing is not more than one or two years old, but poor elsewhere and rough in places. The material used is generally too fine to be durable.

Middleton-Bridgewater highway.—This road is in fair condition, having been recently surfaced nearly its whole length. The surface is rather soft and will wear very rapidly. Some stretches near Bridgewater, which have not been repaired for several years, are in bad condition. Most of the material used is fine gravel carrying a high proportion of sand. The only gravel which is coarse enough to be considered durable occurs at the Annapolis-Lunenburg county line.

Kentville-Chester highway.—The Kentville-South Alton section, which has been surfaced at different times, is in fair condition, except for one mile near Kentville, where the road has a steep grade and is somewhat rough. The South Alton-Forest Home section, and a few short stretches near New Ross and Chester Basin, covered with gravel and sand during the last two years, are in good condition, although the surface is soft. The remainder of the road is in very poor condition, with very bad stretches through poorly drained areas near New Ross Road and Aldersville. The gravel used almost exclusively for covering this road is much too fine for the purpose. Granite boulders are found in large quantity nearly all along this road, and if crushed would produce much better surfacing material.

Windsor-Halifax highway.—From Windsor to east of Newport Corner the road over three years ago was surfaced with gravel from five local deposits, and some sections since repaired with material from the same source. It is now in fair condition in places, poor in others. The gravel west of Ste. Croix is fairly durable, although mixed with much sand, and the gravel found east of Ste. Croix is very soft. The section from east of Newport Corner to Bedford, commenced in 1920 and finished in 1922, is in excellent condition, except the older stretches, which are very uneven. The material used on this road was crushed boulders mixed with fine sand and silt from drift deposits. Gravel is very scarce along this section, while boulder drift deposits are numerous.

Windsor-Truro highway.—The road was examined from Windsor to West Gore only. It has been gravelled during the last two years, except a few short stretches east of Clarksville, and is on the whole in good condition, though very rough on some of the steep grades between Scotch Village and Brooklyn. Gravels from more than ten deposits have been used to surface the road. Most of them are either too soft or too fine to make a durable pavement. More satisfactory material with regard to both size and composition is found at Mosherville station. Here also the deposit carries much sand in places.

Cambridge-Brickton highway.—This road has been gravelled for years and usually maintained in fair condition, except several miles of sand between Berwick and Auburn, which were gravelled for the first time in 1921. This part is now in very good condition, but somewhat soft, the gravel used being too fine. Among the gravels which have been used for road construction, only the following are sufficiently hard and coarse; the gravel at Berwick, West Brooklyn and two miles south of Brickton.

Amherst-Parrsboro highway.—The inspection of this road was made in the early summer, when the Amherst-Southampton section was under construction. From Southampton to Fullerton lake, the road surfaced with local gravel is in good condition, but soft and dusty. Southwest of Westbrook the road passes through a swamp, and is rough and rutted. From Fullerton lake to Parrsboro, the road, surfaced with local gravel and passing through gravelly soil nearly all the way, is in very good condition. The only durable material along the Amherst-Parrsboro highway is found in the Cobequid hills, between Fullerton lake and Lakelands. In order to get a durable surface on the Amherst-Fullerton Lake section, other material than that found along the road will necessarily have to be used.

ROAD MATERIALS

The most common types of suitable road materials encountered are bedrock and gravel.

Since a large proportion of the particles of which the gravels are composed are of the same character as the underlying bedrock, the durability of these materials is closely related. The gravel is in general durable in the hard rock areas, but less so than the bedrock, on account of the presence of a certain proportion of pebbles of other origin. For the same reason, the gravel is soft in the soft rock areas, although less so than the bedrock. Gravel is more plentiful in the hard rock areas, while in the soft rock areas pebble free sand is of more frequent occurrence.

Bedrock

Very hard and tough syenite and syenite porphyry are exposed at several points between Ben Eoin and Big Pond, along the Sydney-St. Peter section of the Truro-Sydney highway. (Table I, Nos. 3, 4). These rocks can be quarried in unlimited quantity.

Extensive trap rock outcrops are found on both slopes of the North Mountains which forms a flat-topped, steep ridge along the north side of the Annapolis valley. (Table I, Nos. 2, 10). This rock is available in any amount in loose blocks forming talus at the foot of cliffs, some within three miles of the main valley road. The same rock is exposed at several places along the Parrsboro-Economy section of the Amherst-Truro highway. This is a very good rock when fresh, but the partly weathered material is much less durable. If used in waterbound macadam construction, weathered rock will rapidly disintegrate into a clay which causes the road surface to be very slippery when wet.

A white, very coarse granite, which underlies a considerable portion of the western half of the mainland, is exposed at numerous places and can be easily obtained in large amounts. It is, however, too coarse and too soft to make a durable road material. The several quarries, southwest of

Middleton, where a sample was taken (Table I, No. 12), show a rather fine crystalline granite, much harder than the ordinary coarse variety commonly found farther south, and quite suitable for roads subjected to ordinary traffic.

The quartzite, or so-called whin rock, and the slate which are found associated together, underlie nearly all the southern and western part of the mainland. Although actual exposures of the rocks are not numerous, the overlying drift carries a tremendous amount of boulders, more particularly of quartzite. The quartzite boulders are nearly all fresh and hard and are a very good material. (Table I, Nos. 6, 13, 14). The schistose quartzite should not be used, as it has not the durability of the massive stone. The slate varies widely in appearance from very soft and shaly to massive and very tough. As seen from most of the exposures and drift boulders, the most common phase is a blue-black, fairly hard stone, with the characteristic slaty structure. It is unsuitable for road surfacing on account of splitting too readily.

The large igneous intrusions found south of the Antigonish-New Glasgow road and north of the Truro-Parrsboro road could furnish a very good road stone, but actual fresh rock exposures in proximity to the main highways are scarce.

Among smaller occurrences of good road material, there is diorite and felsite, exposed at several places along the St. Peter-Hay Cove section of the Sydney-Truro highway, 3 exposures showing fresh and very hard rock; a rock probably altered sandstone, exposed along the same highway just west of Mulgrave (Table I, No. 5); trap, exposed south of Nictaux, along the Middleton-Bridgewater highway; iron ores, of the Torbrook iron mines, the waste of which has been used for road work, (Table I, No. 11); and diabase, exposed along the South Shore at Cherry Hill and Black Point. (See 1921 Report.)

The limestone from the quarry of the British Empire Steel Corporation, near Point Edward, northwest of Sydney, is much superior in quality to other limestones found elsewhere in the province, and is quite suitable for road purposes (Table I, No. 1).

Most of the conglomerates, sandstones (Table I, No. 8), shaly sandstones, and shales are very poor materials, especially those which are geologically classified as belonging to the Carboniferous and Permian formations. Sandstones from older formations are generally harder, of closer texture, and some of them may be used in roads having light traffic, when no better material is available. Such is the case with the sandstone exposed for a distance of several hundred feet, three-fourths of a mile south of McNab Cove, along the St. Peter-Hay Cove section of the Sydney-Truro highway.

The test for determining the cementing power of a rock has not proven so far quite satisfactory, and for that reason too much importance should not be attached to the figures given in the column bearing the heading "Cementing Value." They will serve to indicate only in a general way whether the rock has a high or low cementing power. The test is of value only in cases where a stone is to be used in water-bound macadam surfacing. When bituminous or other artificial binders are used on road surfaces, the cementing power of the rock is of no account.

TABLE I

Results of Tests on Bedrock and other Materials

No.	Location	Type of material	Physical properties							Remarks
			Per cent of wear	French co-efficient of wear	Tough-ness	Hard-ness	Cement-ing value	Specific gravity	Water absorbed lbs. per cu. ft.	
1	B. E. Steel Corp's. quarry, near Pt. Edward.	Limestone.....	3·3	12·1	12	16·7	120	2·71	0·20	Dense stone.
2	B. E. Steel Corp., Sydney	Blast furnace slag...	8·6	4·7	.....	.....	250	2·49	5·40	Not uniform in texture.
3	Southwest of Ben Eoin	Syenite.....	2·1	19·0	25	18·9	48	2·78	0·11	
4	Northeast of Big Pond	Syenite porphyry...	2·2	18·2	27	18·6	50	2·89	0·09	Dense stone
5	West of Mulgrave..	Altered sandstone...	1·5	26·7	18	18·8	32	2·65	0·44	
6	Grosvenor.....	Quartzite.....	2·1	19·0	.....	.....	17	.....	.....	Field boulders
7	James River.....	Felsite and syenite.	1·3	30·8	.....	.....	46	.....	.....	Pebbles from river gravel
8	North of Joggins..	Sandstone.....	6·1	6·6	5	4·5	89	2·54	6·92	Millstone. Very loose texture
9	North Mountain, northwest of Canning	Trap.....	4·1	9·8	10	17·4	110	2·96	1·26	Partly weathered loose rock
10	North Mountain, south of Upper Clarence	Trap.....	2·7	14·8	19	17·9	107	3·00	0·71	
11	Torbrook West....	Iron ore.....	3·3	12·1	20	16·2	32	4·00	1·27	Iron mine dump
12	Southwest of Middleton	Granite.....	2·9	13·8	12	18·7	39	2·70	0·28	
13	East of Liverpool..	Quartzite.....	2·0	20·0	.....	.....	37	.....	.....	Beach boulders
14	West of Tusket.....	Quartzite.....	2·0	20·0	18	18·0	18	2·70	0·47	Field boulders

Gravel

Gravels are fairly common throughout the province, except perhaps in the areas underlain by Carboniferous rocks. They are particularly scarce to the east and south of Sydney, between St. Peters and Hawkesbury, and in the low lands lying between Northumberland strait and the Cobequid hills. Most of the gravels are very fine, and many of them irregularly sorted and interstratified with layers of sand and silt. The coarsest material is, as a rule, found in the upper part of the deposits, and sand mostly in the lower part. Many gravels while composed of very hard particles are too fine to be considered suitable for road purposes. Beach or river gravels are usually coarser and contain a higher percentage of hard pebbles than the corresponding bank gravels, but when applied to a road surface remain loose a long time on account of lack of binder.

Sydney to Truro.—Gravel from a large deposit, one and one-half mile northeast of Blackett Lake, has been extensively used for roads. It is not durable, but, considering the present traffic, it may be suitable for the main road. The beach gravel, available at several points along the shore between East Bay and Ben Eoin, is very hard and coarse, but does not possess sufficient fine material to act as a binder. Away from the shore very fine and hard gravel is found at several places, and can probably be used to advantage on top of beach gravel. A pit recently opened near Cleveland shows fairly hard gravel, although rather fine. This is the only good material found for many miles east and west. The several deposits to the east of Monastery hold probably more sand than gravel. The gravel occurring at Lower South River is too soft for road surfacing. A very large amount has been used for that purpose, since it is the best material found for several miles. Very hard gravel is seen in several pits and road cuts at James River, Marshy Hope, Avondale Station, and Barney River. They are all rather fine, except the river gravel at James River, which is very coarse and bouldery, and carries very little sand. The latter can be used with advantage for the bottom course in either a macadam or gravel road. The main road between New Glasgow and Sutherland River was, in 1921, surfaced with gravel hauled by rail from Avondale Station. In 1922 the road surface was in very good condition, although some of the gravel was still loose. The red gravel east of Telford, which is a loose conglomerate, and the gravels and conglomerates near Alma are too soft for road surfacing purposes. The river gravel found at West River is harder and coarser than any bank gravel occurring within several miles of this place, but is totally devoid of binder. Very durable gravel has been recently excavated at the foot of a knoll west of Lower Mount Thom. Several small road cuts in other deposits show a similar gravel, but with a higher proportion of sand.

Halifax to Shag Harbour.—The gravel is rather scarce between Halifax and Shag Harbour, and when not too fine, is of good quality. The gravels found west of the railway crossing at Mahone and two miles northeast of Bridgewater have given good service on roads for several years. The extensive deposits south of Jordan Falls and south of Sable River Station carry very good gravel, but a large proportion is too fine. Two and one-half miles east of Port Hébert, there is a very coarse and bouldery gravel composed exclusively of hard quartzite, and looks to be very good material.

Middleton to Bridgewater.—The gravel deposit two miles north of Albany Cross, like nearly all the deposits occurring in the granite areas, is very sandy. Although carrying a very high proportion of hard particles, the gravel is too fine to be considered durable. The only deposit of coarse gravel along this road is found between Hastings Junction and Cherryfield, very close to the Annapolis-Lunenburg county line.

Parrsboro to Amherst.—There are several deposits of good gravel near Lakelands. As seen in many road cuts, the deposits carry in places much sand interstratified with the gravel. The gravel at Halfway River station is of fair quality, although irregularly graded. The hauling distance from this deposit to the main road is three-quarters of a mile. A conglomerate, exposed in an excavation south of Aulac, N.B., just past the provincial boundary, has been used to some extent for road surfacing in Nova Scotia. It is composed largely of rounded quartz pebbles in a matrix of loose sandstone. As no suitable road material can be found for many miles south of the border this conglomerate will be extensively used on roads.

A very large railroad pit at Debert station shows a very hard and regularly graded gravel, but having very little fine material to serve as binder. A large amount has been used for road surfacing. Sections of the main road between Shubenacadie and Elmsdale have been surfaced with this gravel and were in excellent condition after two years of service. The sections which have been more recently surfaced show much loose gravel.

A large pit at Berwick shows fairly coarse, hard gravel in the upper nine feet of the deposit, and straight sand farther down.

There is a very large amount of beach gravel available at Digby. It is chiefly composed of trap rock pebbles, of high binding power. A fairly big deposit of bank gravel lying two and one quarter miles south of Digby shows very good material.

Other Materials

Boulder drift deposits, much more common than gravel deposits, are of less importance than the latter as a source of road material on account of their very small percentage of stone of proper size. Their chief interest lies in the fact that they form part of the subsoil on many sections of the main highways, where their composition greatly influences the bearing power and drainage of the road subgrade. When the drift holds a large proportion of hard boulders it may become an important source of material for road surfacing. As a matter of fact, it is the only available material at many points along the south shore. In places the boulders are so numerous that they literally cover the ground. In the case of quartzite boulders alone, there is enough of this stone on the surface available for crushing to pave the South Shore road several times. In localities where it becomes necessary to open pits in the drift deposits, in order to obtain a sufficiently large amount of boulders, no fine material from the pit should be mixed with the crushed boulders to serve as binder in the paving of the road. For that purpose crusher screenings are to be preferred, so as to obtain a homogeneous surface that will be durable and wear evenly.

Slag from the blast furnaces of the British Empire Steel Corporation, at Sydney, has been locally used in road surfacing. The material is soft and irregular in texture, but possesses a very high binding power. The Sydney-Glace Bay road has been partly paved with slag and the surface was very smooth after one year's service, although not very hard. It will probably wear out rapidly, but since the material is available in great quantity and at very low price, frequent re-surfacing can be made at a reasonable cost. It could probably be used to advantage as a binder in a road surface built with hard broken stone or gravel.

TABLE II

Results of Tests on Gravel Samples

No.	Location	Character of gravel						Remarks
		Composition of pebbles			Size			
		Per cent of			Per cent of			
		Durable	Intermediate	Soft	Boulders	Gravel	Sand	
1	N.E. of Blackett lake, 1½ m....	50	10	40	5	60	35	Fels, trap, ss; upper 5′ partly weathered.
2	N.E. of East Bay, 1 mile.....	.....	.....	.....	0	3	97	
3	East Bay, along road to Gillie Lake.	45	30	25	2	60	38	Gr, sy, trap, fels, ss; irregularly graded; sand layers; upper 5′ loamy.
4	N.E. of Ben Eoin, 2 m; along East Bay shore.	100	0	0	1	40	59	Trap, por, sy; all hard; beach gravel.
5	N.E. of Big Pond, 2½ m.....	80	15	5	15	65	20	Por, trap; loose gravel; sand layers.
6	N.E. of St. Peter, 2½ m.....	.....	.....	.....	0	0	100	Intensely weathered sch.
7	N.E. of St. Peter, 2 m.....	0	80	20	1	54	45	Glassy ss; very hard.
8	E. of Cleveland, 2 m.....	0	20	80	1	47	52	Shaly ss; boulder clay; upper 3′ loamy.
9	Near Cleveland.....	40	40	20	10	40	50	Dur: fels, qzte, trap; inter and soft: mostly ss.
10	E. of Tracadie, ¾ m.....	70	10	20	1	57	43	Trap, fels, ss; clayey gravel.
11	Lower South River.....	25	20	55	20	55	25	Dur: trap, gr; inter: trap, ss; soft: ss, sh; streaks of sand; slightly weathered.
12	N.E. of James River, 1½ m., along railway.	90	10	0	1	44	55	Qzte, fels, sy; streaks of clayey gravel.
13	Near Marshy Hope.....	45	40	15	1	51	48	Trap, qzte, fels.
14	S. E. of Avondale sta., ¾ m.; 2 m. N. of main road.	27	30	43	5	55	45	Trap, sy, sh; sand layers.
15	Glenshee.....	18	30	52	20	45	35	75 per cent is sch.
16	E. of Telford, ½ m.....	0	20	80	10	60	30	Fels, ss, gr, sh; partly weathered conglomerate.
17	S. of Alma.....	20	37	43	10	60	30	Fels, qzte, lst, ss; upper 9′ partly weathered.
18	W. of Lower Mt. Thom.....	70	15	15	5	70	25	Sy, trap, ss.
19	N.E. of Mt. Thom, 1½ m.....	0	0	100	0	60	40	Dike of trap; material broken up by weathering agents.
20	Debert sta.....	80	20	0	15	63	22	Gr, sy, por, trap, other ign; regularly graded gravel.
21	Halfway River sta.; ¾ m. E. of main road.	33	45	22	3	72	25	Fels, qzte, gr, ss; sand layers.
22	Opposite Fullerton lake.....	0	5	95	1	57	42	Mostly ss; sand layers.
23	S.W. of Southampton road forks, 1¾ m.	25	30	45	0	55	45	Ss, gr, fels, qzte; sample represents partly weathered gravel.
24	Same as 23.....	.....	.....	.....	0	1	99	Large sand layer in very fine gravel.

25	S. W. of Southampton road forks ½ m	5	5	90	1	28	72	Ign, ss; weathered.
26	Near bridge, along Southampton road.	15	5	80	3	47	50	Mostly ss, some fels and gr; regularly graded.
27	S.E. of Amherst, 3 ½ m	3	0	97	0	30	70	Mostly ss; weathered.
28	S.E. of Aulac, N.B., ¾ m	90	10	0	5	70	25	Qz, fels; partly disintegrated conglomerate.
29	S. of West Gore, 1 m	33	12	55	5	82	13	Sl, ss, trap, qzte, gr; irregularly graded.
30	Mosherville	55	25	20	2	66	32	Trap, fels, ss, sh.
31	Brooklyn	25	35	40	0	70	30	Sl, ss, lst, qzte, gr, fels; deposit mostly sand.
32	N.W. of Albany Cross, 2¼ m	90	10	0	0	45	55	Gr, qzte, trap, fels; deposit mostly sand.
33	Berwick	52	30	18	1	47	53	Fels, trap, gr, ss; sand layers.
34	S. of Digby, 2 m	90	10	0	3	62	35	Gr, qzte, trap; upper 4′ loamy.
35	Near Pleasant Lake	52	43	5	–	60	40	Mostly qzte, well packed; bouldery; looks like boulder sand; irregularly graded.
36	Near Pleasant Lake	60	35	5	5	67	28	Mostly qzte; irregularly sorted; sand and silt layers.
37	N.E. of Barrington, 3 m	90	10	0	10	65	25	Sy, gr, qzte; size varies from place to place; sand layers.
38	S.W. of Birchtown, 3½ m	100	0	0	5	17	78	Gr, qzte; very fine boulder sand.
39	N.W. of Shelburne, ¾ m	100	0	0	10	33	57	Gr, qzte; very fine boulder sand.
40	Shelburne	80	15	5	30	45	25	Qzte, gr, trap; size varies from place to place.
41	N.E. of Shelburne, 2¼ m	80	20	0	–	30	70	Qzte, gr; very fine boulder sand carrying large boulders.
42	S.W. of Jordan Falls, 3 m	67	28	5	3	59	38	Qzte, gr, sch, sh; sand layers.
43	S. of Jordan Falls, 1¼ m	64	21	15	5	50	45	Qzte, gr; sand layers.
44	Near Sable River sta	85	15	0	1	64	35	Qzte; sample represents coarse gravel; deposit largely sand.
45	E. of Port Hebert, 2¼ m	100	0	0	25	50	25	Qzte.
46	S. of Mill Village, 2¼ m	100	0	0	–	20	80	Qzte; fine boulder sand; per cent of boulders varies from place to place.
47	N.E. of Simpson Cor., ⅔ m	65	8	27	1	37	62	Gr, qzte, sl, sh.
48	Bridgewater	30	37	33	1	34	65	Sl, qzte; well compacted; mostly sand.
49	N.E. of Bridgewater, 2 m	35	45	20	5	75	20	Sl, qzte, trap, ss; strongly cemented gravel; per cent of gravel includes many lumps of sand.
50	Near Mahone junction	75	20	5	2	55	43	Qzte, ss, sl; irregularly sorted; sand layers.

TABLE III

Results of Tests on Gravel Samples

No.	Granulometric Analysis													Remarks
	Gravel							Sand						
	Per cent retained on screens							Per cent retained on sieves						Per cent passing 200	
	2½″	2″	1½″	1″	¾″	½″	¼″	8	14	28	48	100	200		
1	5	4	15	21	11	18	26	27	21	22	13	8	3	6	Well graded.
2	.....	.....	.....	.....	.....	4	96	7	17	28	29	14	3	2	Sand sample.
3	3	4	15	23	13	19	23	20	19	28	20	6	3	4	Very good.
4	.....	.....	3	8	14	25	50	18	26	48	8	.....	.....	.....	Beach gravel; lacks binder; suitable for lower course in surfacing.
5	9	13	7	21	15	16	19	38	19	11	7	6	9	10	Very good if boulders removed.
6	.....	.....	.....	.....	.....	.....	.....	11	23	24	20	12	6	4	Weathered schist; unsuitable for roads.
7	.....	5	14	17	13	18	33	27	18	14	9	7	14	11	Well graded, good gravel.
8	5	.....	16	25	14	16	24	16	14	15	14	11	16	14	Boulder clay; very poor material.
9	.....	.....	9	16	10	23	42	27	27	23	9	3	4	7	Very good.
10	.....	21	12	19	12	17	19	13	19	37	19	4	4	4	Should prove a good material.
11	19	14	9	13	9	12	24	43	20	12	5	2	3	15	Too soft and bouldery.
12	.....	5	3	14	16	21	41	21	20	23	17	8	6	5	Very good.
13	.....	9	8	18	12	17	36	36	26	19	7	3	7	2	Sample represents coarser material than average. Good gravel.
14	.....	26	9	14	10	16	25	32	26	25	12	3	1	1	Very good on roads with clay subsoil.
15	.....	9	16	19	12	17	27	26	26	25	12	4	3	4	Soft and bouldery.
16	.....	21	12	19	10	17	21	26	20	19	15	9	5	6	Red conglomerate; very dusty and soft.
17	5	.....	7	16	8	22	42	46	27	16	6	2	1	2	Too soft, due to weathering.
18	.....	4	2	12	13	26	43	64	19	8	4	2	1	2	Very good.
19	.....	.....	11	25	15	18	31	30	21	16	11	8	8	6	Too weathered.
20	9	8	22	18	10	14	19	28	18	22	20	9	2	1	Very good on roads with clay subsoil.
21	9	10	15	15	13	16	22	34	21	19	13	6	4	3	Sample represents coarser than average. Good gravel.
22	7	4	7	16	14	19	33	40	30	15	5	3	3	4	Very soft.
23	.....	.....	.....	21	22	25	32	26	13	16	22	15	5	3	Should give good results on roads.
24	.....	.....	.....	.....	.....	.....	100	.....	.....	1	60	36	2	1	Sand sample.
25	.....	.....	20	7	6	19	48	23	26	18	17	10	3	3	Too soft and fine.
26	.....	.....	6	26	15	20	33	20	19	27	25	4	2	3	Well graded but too soft gravel.
27	.....	.....	16	26	13	18	27	7	4	4	14	28	21	22	Very poor.
28	.....	.....	3	22	17	26	32	45	20	12	9	7	3	4	Conglomerate; suitable for road surfacing.
29	2	7	13	39	21	12	6	15	10	14	23	14	15	9	Irregularly graded; suitable for roads with very light traffic.

30	10	13	10	23	10	15	19	25	15	23	25	6	2	4	Very good.
31	.....	11	11	21	17	19	21	32	19	10	18	7	9	5	Deposit nearly exhausted.
32	.....	.....	8	8	8	19	57	34	28	24	0	2	1	2	Sample represents coarser than average. Much of the gravel is too fine.
33	.....	7	13	17	13	17	33	33	25	25	12	2	1	2	Very good, but has little binder.
34	17	.....	14	18	13	16	22	22	19	22	18	10	5	4	Very good.
35	4	3	27	21	13	14	18	27	15	14	10	5	10	19	Boulder sand; irregularly graded but fairly good.
36	4	13	14	19	13	17	20	30	16	14	14	10	6	10	Same as No. 35.
37	.....	12	28	27	14	14	15	29	15	17	25	8	3	3	Good gravel.
38	.....	13	19	26	8	14	20	11	9	10	12	14	22	22	Boulder sand; per cent of pebbles too low.
39	.....	.....	28	18	11	16	27	6	6	9	12	16	25	26	Same as No. 38.
40	10	10	17	17	10	13	23	26	25	24	13	5	5	2	Very bouldery; has very little binder.
41	.....	17	6	23	14	14	26	11	9	12	15	16	18	19	Fine boulder sand; per cent of pebbles too low.
42	14	19	6	22	11	14	14	18	22	26	15	7	6	6	Very good.
43	16	4	8	19	12	17	24	22	21	26	21	5	3	2	Sample represents coarser than average. Much of the gravel is too fine.
44	6	12	10	27	10	13	22	17	13	20	29	10	4	7	Same as No. 43.
45	8	12	20	23	11	9	17	33	24	17	7	4	4	11	Very bouldery; very good if boulders removed.
46	25	.....	13	9	9	14	30	9	8	9	11	16	21	26	Boulder sand; per cent of pebbles much too low.
47	.....	.....	6	19	13	20	42	20	19	20	21	11	6	3	Very good.
48	.....	.....	21	15	11	18	35	18	17	20	16	10	11	8	Too fine for roads except perhaps on top of a coarse gravel surfacing.
49	27	11	14	15	9	11	13	30	20	17	14	9	5	5	Fairly good.
50	8	.....	22	20	10	16	24	30	31	24	8	2	2	3	Very good.

TABLE IV

Results of Tests on Gravel Samples

No.	Fineness factor	Sand Mortar: 1 cement, 3 sand					Remarks
		Per cent of water used	Tensile strength, Per cent of standard		Compressive strength, Per cent of standard		
			7 days	28 days	7 days	28 days	
1	3·19	15	82	105	72	95	Sand a little dusty and somewhat soft.
2	2·59	14	89	103	79	82	Rather fine.
4	2·54	11	115	108	101	100	Coarse hard sand from beach gravel; good.
5	3·19	16	41	75	24	59	Cracks in briquettes and cylinders, probably due to clay from weathered trap.
7	2·74	15	99	114	95	104	Very irregularly graded sand.
11	3·43	14	106	130	99	103	Much fine passing 200 mesh; fair.
14	3·66	12	166	147	149	152	Very good.
15	3·37	13	171	145	136	140	Very good, but a little dusty.
16	3·06	16	97	106	68	75	Very poor, due to metallic oxides as impurities.
17	4·00	13	229	197	174	177	Coarse sand; very good.
20	3·27	13	76	87	97	121	Coarse sharp sand; poor on account of rust.
24	1·59	17	65	84	43	54	Clean sand; too fine to be suitable.
26	3·11	13	100	112	88	105	Fair.
30	3·10	12	119	122	118	119	Coarse quartz sand; very good.
32	3·74	11	94	78	112	103	Sand from disintegrated granite; very poor.
34	2·98	12	111	79	123	135	Sample from near surface includes a little loam; good.
36	2·94	13	95	110	104	117	Very irregularly graded and too fine.
37	3·14	13	58	59	65	83	Sand from disintegrated granite; very poor.
38	1·21	18	54	57	49	68	Too much fine sand and silt, mostly quartz.
40	3·33	13	119	117	115	135	Good.
42	2·93	13	62	54	63	89	Clean, but too fine sand.
43	3·19	13	139	140	135	146	Very good.
44	2·65	12	87	83	116	137	Clean quartz sand, somewhat fine.
45	3·30	12	103	104	112	122	Fair.
47	2·89	12	93	94	94	93	Some soft shale and slate grains; very poor.
50	3·64	15	0	56	0	44	Cement had not set after 7 days, probably due to too much rust in the sand.

Explanation of Tables II and III

The proportions given for the pebble composition can be taken as representing the average composition of each deposit, but the same cannot be said about the classification as to size. Great care was taken in collecting samples that would represent the pit average as to the size of the particles, but when a thick sand layer intervened it was not included in the sample. In most cases the samples comprised the gravelly parts of the bank and thin layers of sand and silt occurring between gravel layers. In some cases the proportion of sand to gravel is so variable in different parts of the deposit that care must be exercised in applying the figures of percentage given for the average.

Abbreviations used in remarks

dur	durable
fels	felsite
gr	granite
ign	igneous
inter	intermediate
lst	limestone
por	porphyry
qz	quartz
qzte	quartzite
sch	schist
sh	shale
sl	slate
ss	sandstone
sy	syenite

Explanation of Table IV

The standard test referred to in Table IV is made on test pieces composed of 1 part cement, 3 parts Ottawa standard sand. Samples testing at least 100 per cent of the standard are considered suitable material for concrete works. All samples were taken from gravel and sand deposits, except No. 24, which is from a straight sand deposit, and No. 38, which is from a drift deposit, largely composed of fine sand and silt. As seen in the table, the fineness factor of the latter two samples is much below that of the gravels, which accounts for their low tensile and compressive strengths.

APPENDIX

INTERIM REPORT OF THE JOINT PEAT COMMITTEE

(December 5, 1922)

B. F. Haanel
Secretary to the Committee

The original objective which the Joint Peat Committee had in 1918, the year of its appointment, was the testing of peat machines under commercial conditions, in order to determine if a type suitable for manufacturing peat fuel on a commercial basis could be developed. By a process of elimination two types were selected, the well known Anrep peat machine, largely used in European countries—in the smaller sizes—and the Moore machine. The latter, while not fully developed, appeared to possess so many advantages that the Federal government, before the appointment of the Peat Committee, decided it should be given a trial.

On account of the precedence given to war contracts, the two machines were not delivered at the Alfred bog, Ontario, until the spring of 1919, and their erection was only completed in time to enable a short mechanical test to be conducted before cold weather set in. It was not, therefore, until the spring and summer of 1920 that manufacturing operations were actually begun.

Up to this time the Committee was under the impression that its work would be completed at the conclusion of 1920, and that sufficient data would be available to enable an opinion to be expressed concerning the commercial possibilities of the two machines. The results of that season's operations, however, disclosed inherent weaknesses in both plants, and the Committee was unable to approve either type for commercial work. Recognizing the great value to the Dominion at large, of a healthy peat development, the Committee recommended that a new combination be effected embodying the best features of the two types which had been tried. This recommendation was approved by the two governments and funds were allotted for carrying out the construction and operation of the new combination. The construction was completed late in 1921 and operations were concluded in 1922.

DEVELOPMENT OF PEAT MACHINERY

The success of any such machine depends largely upon the elimination of manual labour to the greatest possible extent and on the production of a manufacturing unit of large capacity at the lowest possible cost commensurate with good service.

Attempts to meet these two conditions led to the trial of many mechanical devices which heretofore have not been used in this connection. The most noteworthy features involved were the adoption of caterpillar

elements and the substitution of a portable belt conveyer and automatic spreading machinery for the track system and field presses formerly employed. The development of these and other mechanical features proved difficult, and occasioned the expenditure of much time. While the machine has not actually been constructed according to the design which the Committee considers would be reasonably free from defects, the problems involved have, in their opinion, been thoroughly investigated, and drawings for a complete standard equipment with power plant, pulverizing unit, conveyer and spreading system are now under way.

NECESSITY FOR FURTHER GOVERNMENT OPERATION

While the Committee has completed research work requisite to the development of a successful peat manufacturing plant, and is able to place at the disposal of those interested construction data and operating statistics, it is of opinion that the establishment of a peat industry is of such importance to Canada, and especially the province of Ontario, that the governments should undertake the construction and operation of a perfected plant until its possibilities are fully demonstrated.

COST OF MANUFACTURING FUEL WITH A COMBINATION PLANT

The Committee after careful consideration arbitrarily fixed the price f.o.b. cars peat plant siding, at which peat fuel could in its opinion be sold for consumption, within a 100-mile radius and still compete with anthracite coal. The price was $5 per ton of standard peat fuel containing 30 per cent moisture, and comprises the following items:—

1. Production cost.
2. Overhead charges.
3. Manufacturer's profit.

The production cost depends mainly upon the cost of the raw material, the prevailing price of labour, and number of employees, seasonal variations, and the quantity of saleable fuel produced per unit per working season.

Overhead charges include interest on capital invested, depreciation and administration.

Manufacturer's profit as would be arrived at by the investor.

The figures employed in this report are considered fair by the Committee, but those who desire to engage in such an enterprise would naturally estimate their own overhead charges and profit.

In order to keep items 1 and 2 sufficiently low to enable a fair manufacturer's profit to be realized and still bring the selling price of peat f.o.b. cars within the limit of $5, it was estimated that the minimum capacity of the unit must be 10,000 short tons of saleable standard peat fuel for a working season of one hundred days. This type of machine should be capable of producing a sufficiently high hourly average of peat fuel throughout an entire season, irrespective of seasonal variations and delays due to breakdowns or other causes, to enable the 10,000 ton figure to be realized. This we believe could be accomplished by constructing the complete unit sufficiently large to produce a maximum capacity considerably in excess of the minimum required.

The improved plant, exclusive of harvesting equipment, consists of four components:—

1. Excavating.
2. Macerating.
3. Distributing and spreading.
4. Power.

The excavating element proved of sufficient capacity to meet all requirements, and the distributing and spreading system was sufficient to deliver and spread the maximum quantity of peat excavated. The macerating element, however, failed to deliver the required quantity with the power available. This was due to the adoption of a new type of macerator, a Swing Hammer Pulverizer, which it was considered would prove far more reliable, cheaper and more efficient than the Anrep macerator heretofore employed. This machine had not before been employed for macerating peat, and, therefore, its capacity and the power required to operate it for such purpose were not known. A portion of the operating season had to be devoted to experimentation with and calibration of this unit.

On account of the lateness of the season, and lack of funds, the Committee could not possibly consider the purchase of a power plant sufficient to operate the combined plant to full capacity. The power plant used was improvised, comprising two boilers already on the bog, the wasteful steam engines which formed part of the two plants previously experimented with, and a high speed engine which was rented for the season. The two boilers were rated at 80 horsepower each and the engineer of the Committee had reason to believe that practically 160 horsepower would be available. A test, however, disclosed that only 110 horsepower could be developed, while the minimum required to operate the full capacity was estimated to be 150 horsepower. With 150 horsepower and a larger macerator the full capacity of the plant could have been delivered. Since both these elements are standard equipment and can be obtained in various sizes on the market, the ultimate capacity of the plant can readily be obtained by simply substituting a macerator and power plant of greater capacity.

COSTS

Based on the performance of the experimental combination plant and the estimated cost of an entirely new and remodelled plant, complete with an efficient power unit and larger macerator, the production and other costs of the finished peat per ton are estimated to be:—

	10 hour day	20 hour day
Production costs	$2 00	$2 00
Overhead charges	2 48	1 50

or a total cost of finished peat fuel, on board cars at siding of plant, of $4.48 for a ten hour day for a season of 100 days, or $3.50 for a twenty hour day during the same season. A total production of saleable fuel in the first case is 10,000 short tons and in the second, 20,000 short tons. Since, however, the overhead costs mount rapidly as the production decreases the Committee recommended that plants of this type should be

operated for twenty hours per day. The necessity for increasing the length of the working season by operating twenty hours per day will be readily appreciated, when it is realized that the considerable investment represented in plant is, under the most favourable conditions, lying idle practically two-thirds of the year. Thus the overhead would be distributed over the production of only one-third of the year, greatly increasing the cost. Working two shifts is equivalent to operating ten hours a day for 200 days. Night operation with this combined plant is entirely feasible; it was not the case with either of the others.

The foregoing costs are below that arbitrarily set by the Committee as the maximum price at which peat could be sold f.o.b. at plant and still compete with coal for domestic purposes. The most notable feature is the marked reduction in the total cost due to reduction in overhead, by operating for a season of two thousand hours instead of one thousand.

The important reductions in the final total cost will be attained by many improvements permitting of substantial saving in the construction of a new plant, and by the employment of a self-contained Diesel engine electric power plant. By using Diesel engines and electric generators, and operating all drives on the plant with electric motors, it is estimated that a saving of 85 cents per ton of saleable peat fuel will be realized, over that possible with the inefficient outfit employed this season. These figures assume the employment of this or some other equally efficient power plant. For a plant producing ten thousand tons, in 1,000 hours, a power equipment of 200 horsepower will be required. The following table recites the estimated costs of building an entirely new plant, with power, harvesting, loading equipment and other accessories. Overhead and administration charges are also shown.

TABLE OF NEW PLANT COSTS

	Capital cost	Interest	Depreciation	Administration
Power plant	$25,000		10%—$2,500	10% on $100,000 for 20 hours daily operation
Peat plant	35,000	7%	20%—7,000	
Harvesting equipment	25,000		14%— 3,500	or
Buildings, equipment and miscellaneous	5,000		10%— 500	10% on $50,000 for 10 hours daily operation
Total	$90,000	6,300	13,500	$10,000 or $5,000.

MARKET

A good market has been developed for peat within the economical shipping radius of the Alfred bog. The demand is growing. The success of such a plant depends in a marked measure, however, upon the site chosen for manufacturing in its relation to points of consumption. This should be selected with a view to eliminating long rail hauls. The economic

limit depends not only upon freight rates, but on the price of other available competitive fuel. For example, the cost of anthracite to consumers has an effect upon the price at which peat fuel can be sold at the same point. It is the opinion of the Committee, that the price of peat fuel delivered to consumers should be in the neighbourhood of $10 per ton if the bogs are strategically situated with respect to transportation facilities and within 100 miles of the market.

PLANT NO. 3, SMALL PEAT PLANT

When Plant No. 3, the small plant, was first designed, an attempt was made to combine the excavating and macerating units in a single machine, simple in design and inexpensive. Until the end of the season of 1921 efforts were made to perfect the original design, but the results were not satisfactory. Difficulties with the combined excavator and macerator were not completely overcome, and it was decided that other means of macerating must be found before this type of machine could be considered practical from a commercial standpoint. About this time the possibilities of the Swing Hammer Pulverizer were brought to the notice of the Committee and nothing further was done with No. 3 until this machine had been tested.

The Swing Hammer Pulverizer proved to be so superior to the type of macerator formerly used that it was decided to include it in the rebuilding of No. 3. This was carried out during the 1922 season. Construction, however, was only completed after the normal working season, and, therefore, no sustained demonstration was possible. The troubles which developed in the first design were overcome. The capacity of No. 3 plant should be not less than 1½ tons standard fuel per hour and in regular operation might exceed this. The cost of construction cannot be given at this time, as new drawings have to be prepared. It is, however, expected that it will not exceed $5,000. A full description of the redesigned machine, with a discussion of its capabilities, and accurate figures of its cost, will be given in the final report of the Committee. Three men should be sufficient to operate it.

TORONTO, December 5, 1922.

(Sgd.) A. A. COLE,
Chairman,
Peat Committee.

(Sgd.) B. F. HAANEL,
Secretary,
Peat Committee.

(Sgd.) R. C. HARRIS,
Member,
Peat Committee.

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